PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 17.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

1915.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

LIST OF OFFICERS, 1915.

President:

ROBERT C. STICHT, Queenstown, Tas.

Vice-Presidents:

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D. B. WATERS, Dunedin, N.Z.

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R. S. BLACK	Western Australia
H. W. GEPP	New South Wales
RICHARD HAMILTON	Western Australia
STANLEY B. HUNTER	Victoria
A. S. KENYON	Victoria
A. H. MERRIN	Victoria
JAMES HEBBARD	New South Wales
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WILLIAM POOLE	New South Wales
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C. F. COURTNEY	New South Wales
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D. L. STIRLING, Melbourne, Vic.

LOCAL CORRESPONDENTS:

South Australia	JAMES P. WOOD, Adelaide.
New South Wales	F. DANVERS POWER, Sydney.
New Zealand	D. B. WATERS, Dunedin.
Tasmania	RUSSELL M. MURRAY, Mount Lyell.
North Queensland	MURRAY RUSSELL, Cloncurry.

BRANCH SECRETARIES:

Kalgoorlie	F. W. R. GODDEN, Ivanhoe Gold Corporation, Boulder, W.A.
Broken Hill	V. A. HAIGH, Central Mine, Broken Hill, N.S.W.
Mount Morgan	B. G. PATTERSON, Mount Morgan, Q.

The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 SWANSTON STREET, MELBOURNE, VICTORIA.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE INSTITUTE.

(Abstract).

A Special Meeting of the Institute was held at the rooms, 57-59 Swanston-street, Melbourne, on Friday, 29th January, at 12.15 p.m.

The meeting was called for the purpose of the examination of ballots for the alteration of Rule VII.

Mr. H. H. Schlapp occupied the chair and appointed Messrs. S. N. Rodda, K. C. Church and J. E. Hamilton scrutineers.

The result of the ballot was announced by the Chairman who declared the Rules altered as follows :—

RULE VII.

Clause 2. *Delete* "six" *insert* "seven."

Clause 6. After the word "death" *insert* the words "cessation of Membership."

Clause 7. *Delete* "next ensuing Annual Meeting," *insert* "expiry of the term for which the officer replaced by him shall have or has been elected."

Delete "not render him ineligible for re-election," *insert* "be approved by a majority of votes at a ballot to be taken at the next ensuing Annual Meeting and provided also that notice of such appointment shall have been posted to each Member not less than five weeks before the date of such Annual Meeting. Casual vacancies filled later than six weeks from the date of the next ensuing Annual Meeting shall be balloted for at the second ensuing Annual Meeting."

Annual Meeting,

MELBOURNE, 29TH JANUARY, 1915.

IN THE INSTITUTE ROOMS, 57-59 SWANSTON STREET, AT 12.30 P.M.

Mr. H. H. Schlapp occupied the chair.

The Secretary read the notice convening the meeting.

Minutes of the First Ordinary Meeting, 1914, and Special Meeting held on 22nd September, 1914, were taken as read and confirmed.

The report of the Council and Balance Sheet for 1914, were read and adopted.

The result of the ballot for the election of officers to fill the vacancies was reported by the scrutineers to be as follows :—

President - -	ROBT. C. STICHT, Tas.
Vice-President -	JAMES HORSBURGH, Q.
Council - -	E. G. BANKS, N.Z.
	W. H. CORBOULD, Q.
	C. F. COURTNEY, N.S.W.
	H. LIPSON HANCOCK, S.A.
	H. HERMAN, Vic.
	ERNEST W. SKEATS, Vic.
	JOHN W. MOULE, Q.

The meeting then closed.

Report of the Council for 1914.

TO THE MEMBERS.

The Council has pleasure in submitting the following report of the Proceedings of the Institute for the year 1914.

The number of Members on the Register is now 735, made up as follows:—Honorary Member, 1; Fellow, 1; Life-Members, 5; Members, 445; Associate Members, 194; Students, 89—a decrease of 10, after allowing for deaths, resignations, and removals.

Twenty-three Members resigned during the year; 31 were removed from the register for non-payment of subscriptions.

Three meetings of the Institute were held—the Annual Meeting at Melbourne on 29th January, the First Ordinary Meeting, 1914, during May, and a Special Meeting at Melbourne on 22nd September.

The First Ordinary Meeting included excursions to the Walhalla goldfield and to the State coal mine at Wonthaggi, besides visits to places of engineering interest in and around Melbourne. An account of this meeting was published in Proceedings, No. 14. The success of the meeting was largely due to the co-operation and assistance of the managers and staffs of the mines and works visited, to the Victorian Railway Department, to the Melbourne Harbour Trust, to the Shire Councils of Walhalla and Wonthaggi, and to those gentlemen who placed their motor-cars at the disposal of the visitors on 23rd May. To one and all the Council accords its hearty thanks.

The First Ordinary Meeting, 1915, will be held at Mount Lyell, Tasmania. Members will leave Melbourne by the s.s. *Oonah* on Friday, 5th March, returning on 14th of the same month. Ten years have elapsed since the last Institute visit to this famous mine, and the alterations and improvements during that time have been in keeping with the progressive policy of the Mount Lyell Mining and Railway Co. Members are strongly advised to take advantage of the special opportunity which this meeting affords. The Mount Lyell Co. has already taken special steps to make the occasion particularly attractive.

The Broken Hill Branch more than maintains the interest of members in the Institute in that particular district. The Auckland Provincial Branch, owing to various causes, chief amongst which was the decline of mining in that district, was obliged to disband.

Publications issued during the year were :—Proceedings (new series), Nos. 12, 13, 14, 15, and 16. A complete index of all papers published by the Institute has been prepared, and will be printed as soon as possible.

Owing to the large increase in membership of the Institute during recent years, and in order that each State should have, as nearly as possible, a representation on the Council proportionate to its number of members, an increase in the Council was suggested and approved. Rule VII. was thereupon altered by ballot at a Special Meeting held on 22nd September to provide for twenty-one members of Council instead of eighteen. Further consequential alterations have been recommended by the Council. These alterations will be the subject of a ballot at a Special Meeting on 29th January.

During the year Mr. F. G. Wilson was appointed local secretary to the Broken Hill Branch, and Mr. F. W. R. Godden local secretary to the Kalgoorlie Branch. The thanks of the Council have been accorded to Messrs. J. C. Coldham and J. L. Connor for having so long and ably filled these respective offices.

What may be considered the most important event in the history of science in Australia was the visit of the British Association for the Advancement of Science in August. The Council of the Institute took a prominent part in the arrangements for this visit, and in the carrying out of a very elaborate programme.

The International Engineering Congress, to be held at San Francisco on 20th to 25th September, 1915, under the auspices of all the leading societies and institutes in America, should be an occasion of great importance to the engineering profession. Your Council has given its assistance in the dissemination of the literature of the Congress. Over 200 papers are listed for reading and discussion.

The reorganization of the Melbourne Technological Museum has had further assistance from the Council during the year, and a report from the Executive Committee has now been received.

The advisability of the incorporation of the Institute is a matter which has, on various occasions, been discussed in a general way. Towards the end of the year the matter was materially advanced by the Broken Hill Branch by forwarding to the Council a direct proposal for incorporation accompanied by a draft copy of Memorandum and Articles of Association. The scheme submitted is a very comprehensive one, and, if carried into effect, would mean extending the functions of the Institute very considerably. The matter is now before each Member of the Council, and an announcement will be made at a later date.

The financial position of the Institute is sound, notwithstanding the present unsettled position in the financial world and its effect on mining. The expenditure on publications exceeded the estimate. This was due to the extraordinary size of No. 14 of Proceedings; but it was felt that the character of the papers published in that number fully warranted the extra expenditure.

AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

Balance Sheet, 31st December, 1914.

LIABILITIES.

	£	s.	d.	£	s.	d.
Sundry Creditors				£152	12	3
Subscriptions paid in advance				28	9	6
Accumulated Fund—						
Balance at 31st Dec., 1913	£375	13	1			
Add Balance transferred from Revenue Account	228	18	5			
				604	11	6
				£785	13	3

ASSETS.

	£	s.	d.	£	s.	d.
Cash in Bank of Australasia	£19	7	7			
Cash in Savings Bank of Victoria	36	4	8			
Cash in hand	15	15	6			
				£ 71	7	9
Fixed Deposit in Bank of Australasia ...				200	0	0
Outstanding Subscriptions, 1912, 1913	72	19	3			
Outstanding Subscriptions, 1914	375	8	6			
				448	7	9
Sundry Debtors for Advts. ...	25	0	0			
Suspense Account — Cash in hands of New Zealand Branch	2	16	3			
				27	16	3
Office Furniture, Melbourne and Broken Hill Branch ...	42	6	0			
Less Depreciation	4	4	6			
				38	1	6
				£785	13	3

Melbourne, 27th January, 1915. Audited and found correct. B. H. Oxlade, A.C.P.A., Auditor.

Statement of Receipts and Expenditure for the Year ended 31st December, 1914.

RECEIPTS.

	£	s.	d.	£	s.	d.
To Bank Balance, 1913 ...	£24	18	3			
Cash in Hand	25	6	6			
Cash in Savings Bank ...	31	15	10			
				£82	0	7
Members' Subs., 1911 ...	2	2	0			
" " 1912 ...	18	18	6			
" " 1913 ...	88	18	0			
" " 1914 ...	961	15	0			
" " 1915 ...	26	18	0			
				1098	11	6
Advertisments				62	10	0
Sale of Publications				10	7	3
Interest				4	8	10
Entrance Fees				70	7	0
Sundries				24	15	9
Subscribers				1	11	6
				£1354	12	5

EXPENDITURE.

	£	s.	d.	£	s.	d.
By Postages, Telegrams, &c.	£64	9	6			
Printing & Stationery ...	22	9	0			
Salaries	250	0	0			
Rent	36	0	0			
				£372	18	6
Sundries				39	10	7
Meeting Expenses				86	12	3
Branches				80	0	0
Proceedings				504	3	4
Fixed Deposit—Bank of Australasia...				200	0	0
Cash in Bank of Australasia	19	17	7			
Cash in Savings Bank of Victoria	36	4	8			
Cash in Hand	15	15	6			
				71	7	9
				£1354	12	5

Revenue Account for the Year ended 31st December, 1914.

Date	Particulars	£	s.	d.	£	s.	d.
1914.							
Dec. 31.	To Publications	£640	10	1			
	Meeting Expenses	86	12	3			
	Branches ...	65	4	3			
	Stationery & Printing ...	26	14	6			
	Postages ...	64	9	6			
	Salaries ...	250	0	0			
	Rent ..	48	0	0			
	Sundries ...	29	10	7			
					£1211	1	2
	Bad Debts—Subscriptions				181	18	0
	Depreciation				4	4	6
					1397	3	8
	Accumulated Fund ...				228	18	5
					£1626	2	1

Date	Particulars	£	s.	d.
1913.				
Dec. 31.	By Advertisements and Sales ...	£70	7	3
	Subscriptions, 1914	1479	7	6
	Interest	4	8	10
	Entrance Fees	70	7	0
	Subscriptions, Subscribers ...	1	11	6
		£1626	2	1

Melbourne, 27th January, 1915. Audited and found correct. B. H. Oxlade, A.C.P.A., *Auditor.*

First Ordinary Meeting, 1915, Queenstown, Tas.

5th March to 14th March.

Friday, 5th March.

The programme issued in connection with the First Ordinary Meeting, 1915, attracted an exceptionally large number of members from the various States and New Zealand. The party, which left Melbourne by the s.s. *Oonah* on 5th March, totalled 62.

Saturday, 6th March.

On arrival at Burnie, Tas., early on Saturday morning, a number of Tasmanian members joined the party. At Guildford Junction, Rosebery, and Zeehan further additions were made. On reaching Queenstown the visitors numbered 80 strong. Early breakfast was provided at Burnie, morning tea at Guildford Junction, lunch at Zeehan, and afternoon tea at Regatta Point, where members were met by the President (Mr. R. C. Sticht), and the local committee. Queenstown was reached shortly after 6 o'clock in the evening.

Sunday, 7th March.

At 9 a.m. the party left Queenstown in trucks, specially fitted, and drawn by a Krauss locomotive. These trains were at the disposal of members during their stay in Queenstown. At the Queenstown end Howard's Plain was ascended by haulage. Six miles of 2-ft. gauge tramway was then traversed, and the power station was reached, where light luncheon was provided by the Mount Lyell Mining and Railway Company, and was available throughout the day. The party, in batches of 20, was taken to Lake Margaret *via* the haulage line. An inspection was also made of the power station. Queenstown was reached, on return, at about 5.30 p.m.

Monday, 8th March.

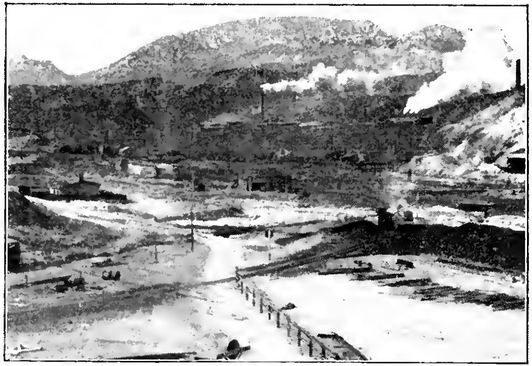
A start was made from Queenstown railway station at 9 a.m. for visits to the Mount Lyell and North Lyell mines. On arrival at the foot of the haulage the visitors divided into two equal parties—one for the Mount Lyell mine and one for the North Lyell mine. At the top of the haulage the Mount Lyell party inspected the open-cut and surface workings, and then proceeded underground through No. 5 tunnel. The North Lyell party was conveyed by tram from the top of the haulage to the North Mount Lyell mine, stopping to inspect the timber aerial and yard *en route.* An inspection was made of the surface workings, and subsequently of the underground workings. A departure was made for Queenstown at 5 p.m.

At 8.30 p.m. the President entertained members at a smoke night at the Masonic Hall. The function was a most enjoyable one. The principal toasts honoured were:—"The King and Empire," given by the President; "Our Guests" was also given by the President, and acknowledged by Mr. Jas. Hebbard; "The Mount Lyell Mining and Railway Co. Ltd. and the Mining Industry," was proposed by Professor D. B. Waters, and acknowledged by Messrs. E. Carus Driffield (superintendent railways), R. M. Murray (engineer-in-charge), for the company, and by Mr. J. S. Munro for the mining industry: "The President," was given by Professor E. W. Skeats, and acknowledged by Mr. R. C. Sticht. A musical programme greatly added to the enjoyment of the evening.

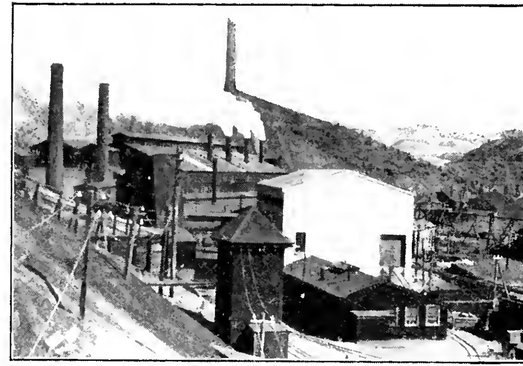
Tuesday, 9th March.

At 9 a.m. members were conveyed to the smelters. On arrival at the reduction works, 710-ft. level, the party was divided into sub-parties, each in charge of a guide, and conducted through the works. Those who had not inspected the Mount Lyell and North Mount Lyell mines on the previous day were afforded an opportunity for doing so. The itinerary at the mines was the same as Monday, except as to times.

First Ordinary Meeting, 1915—Members Leaving Queenstown.



General View of Works, Mt. Lyell.



Blast Furnace Plant, Mt. Lyell.

Steel Pipe-Line and Power House, Lake Margaret, Mt. Lyell.

Wood Pipe-Line, Lake Margaret, Mt. Lyell.

Several of the party took advantage of an excursion which had been arranged to the King River Gorge Park. For this visit the train left Queenstown at 1.30 p.m., arriving at Queenstown, on return, at 6.35 p.m.

At 8 p.m. members assembled at the Masonic Hall to attend the business meeting and the reading and discussion of papers. The President occupied the chair.

Minutes of the Special Meeting and Annual Meeting, held on 29th January, were read and confirmed.

Mr. A. S. Kenyon briefly explained the scheme of incorporation of the Institute as submitted by the Broken Hill Branch, and it was resolved that the scheme, as suggested, together with suggestions to date thereon, be printed and submitted to members. Professor D. B. Waters proposed, and Mr. Jas. Hebbard seconded, "That it be a recommendation to the Council that the First Ordinary Meeting, 1916, be held in the South Island of New Zealand." The motion was carried.

Mr. Jas. Hebbard proposed, and Mr. A. S. Lilburne seconded, "That it be a recommendation to the Council that a donation of at least £50 be made by the Institute to the Belgian Relief Fund." The motion was carried.

Mr. Jas. Hebbard, on behalf of the Broken Hill Branch, congratulated members on the great success of the present meeting.

Votes of thanks to the Mount Lyell Mining and Railway Co., the Tasmanian Government Railways, and to all those who contributed to the success of the meeting, were accorded by acclamation.

The following papers were then read and discussed :—

"Mining Methods at Mount Lyell." By R. M. Murray.

"Some Geological Observations at Mount Lyell." By Hartwell Conder.

During the evening each member was presented by the Mount Lyell Mining and Railway Company with a beautiful little souvenir in the form of a booklet containing a summary of points of interest at Mount Lyell. The booklet also contained eighteen original photographs of special interest.

Wednesday, 10th March.

On Wednesday members were the guests of the Mount Lyell Mining and Railway Company on an excursion to the famous Gordon River. A start was made from Queenstown at 6 a.m. On arrival at Linda the North Lyell train conveyed the party to Kelly Basin, on Macquarie Harbour, a distance of 28 miles. At Kelly Basin two small steamers were boarded, and, with a large barge in tow, steamed past Settlement Island and for a distance of 18 miles up the Gordon River, "a noble waterway guarded by densely wooded heights of entrancing loveliness." Here the steamers were moored alongside the barge, whereon a splendid lunch was done full justice to. A return was then made, and Queenstown reached at 10 o'clock.

Thursday, 11th March.

A departure was made from Queenstown at 9 a.m. by special train for Zeehan. After lunch a special train, provided, free of cost, by the Tasmanian Government Railway Department, conveyed the party to Renison Bell, on the North Dundas tinfield. Here an inspection was made of the Renison Bell and Dreadnought–Boulder mines, under the guidance of Mr. James B. Scott, manager. A return was then made to Zeehan. The local arrangements were in the capable hands of Mr. A. D. Sligo, of Zeehan.

In the evening a visit was paid to the Zeehan School of Mines, when Mr. D. V. Allen, the director, took members in hand. In addition to the general equipment of the school, the very fine collection of minerals belonging to the school was much admired.

Friday, 12th March.

At 8.30 a.m. a start was made for Williamsford in the North-East Dundas tram. The tram was drawn by a powerful Garrett locomotive, and wound its way with ease up and up the mountain sides, past the famous Montezuma Falls, to a height of 1500 ft. Williamsford was reached in good time, and there the party was met by Mr. C. Moxon (of the Hercules mine) and Mr. G. Barker (of the Tasmania Copper and Primrose mines).

The remainder of the journey ($4\frac{1}{2}$ miles) to Rosebery was accomplished on foot by most of the party, carts being provided for the ladies and older members. At Rosebery light luncheon was done full justice to. Here an inspection of the Tasmania Copper mine and the Primrose mine was made. The up train was boarded at Primrose siding. At Guildford Junction afternoon tea was appreciated, and a special train conveyed members to Waratah, where the party was received by Mr. J. D. Millen, general manager of the Mount Bischoff Co. In the evening members and a number of townsfolk were right royally entertained by the Mount Bischoff Co. at a social evening and supper. Mr. Millen, acting as host, gave a hearty welcome to the visitors, which was suitably acknowledged by Professor D. B. Waters.

Saturday, 13th March.

An inspection of the Mount Bischoff mine and works was made on Saturday morning, and at 3 p.m. a departure was made for Burnie. On arrival at Burnie, at 6 o'clock, Mr. W. Reid Bell (engineer) and Warden Smithies (of the Marine Board of Burnie and Table Cape) conducted the party over the new harbour works, when a 60-ton block was conveyed from the shore and lowered into position. This proved an interesting finale to one of the most successful meetings in the history of the Institute.

Members for the mainland left by the s.s. *Oonah*, and, after a calm passage across the Strait, arrived in Melbourne at 2 o'clock on Sunday afternoon.

OF THE COUNCIL.

(Abstract).

FEBRUARY 18TH, 1915—4.30 P.M.

A Special Meeting of the Council was held on February 18th to welcome the President (Mr. R. C. Sticht) on his return from America and to receive a progress report of arrangements in connection with the First Ordinary Meeting, 1915.

After a welcome had been extended to the President the Secretary reported developments in connection with the First Ordinary Meeting.

MARCH 1ST, 1915—4.30 P.M.

It was resolved that an Executive Committee be appointed for 1915, consisting of all members of the Council resident, or, for the time, being in Melbourne.

OF THE EXECUTIVE COMMITTEE.

(Abstract).

DECEMBER 21ST, 1914—430 P.M.

The Secretary's report was presented and accounts to the amount of £59, including rent £12 and Broken Hill Branch £25, were passed for payment.

Nomination of Mr. F. M. Murdoch as a Member was approved.

Mr. James P. Wood of Adelaide, was appointed Local Correspondent for South Australia.

Authority was given for the issue of a programme of the First Ordinary Meeting, 1915.

Written communications from Councillors on the subject of Incorporation were submitted.

JANUARY 27TH, 1915—4.30 P.M.

The Secretary's report was presented and accounts to the amount of £93, including printing £50, were passed for payment.

Nominations as follows were approved :—Members—Messrs. F. W. R. Godden and Hartwell Conder. Associate Member—Mr. Valentine A. Corstophan.

Resignations of Messrs. Albert Morris, Leonard G. Sewell and B. B. Fabian were accepted.

Resolved that all members who had been accepted for active service with the Commonwealth Expeditionary Forces be exempted from payment of their subscriptions during their period of service.

The following were admitted to the Institute :—As Members—Messrs. W. Baragwanath, Robert Hasson and Frederick M. Murdoch. As Associate Members—Messrs. H. R. Allen, C. S. Barber, A. G. Campbell, D. C. MacGruer, A. E. Powell, N. Sturzaker, A. Williams and R. J. Winters.

A Paper by Mr. J. H. Ledeboer on "The Application of Surface Combustion" was referred to the Publication Committee.

Draft of the Annual Report of Council for 1914, was submitted and approved.

Mr. B. H. Oxlade was appointed Auditor of the Institute accounts for 1914.

Mr. H. H. Schlapp was appointed Chairman and Mr. C. F. Courtney Vice-Chairman of the Executive Committee for 1915.

MARCH 1ST, 1915—4.35 P.M.

The Secretary's report was presented and accounts to the amount of £149, including publications £90, were passed for payment.

Nominations as follows were approved :—Member—Mr. Leslie V. Waterhouse. Associate Members—Messrs. John K. Campbell and Brian W. Lennon.

Sub-Committees for 1915 were appointed.

Resolved that the scheme of Incorporation submitted by the Broken Hill Branch be published in the Proceedings, together with a brief statement showing what the proposal involves.

The Secretary was instructed to proceed to Mt. Lyell to conduct the business, &c. of the First Ordinary Meeting, 1915.

A Paper by Professor D. B. Waters entitled "Mining Education in Australasia," was referred to the Publication Committee.

Other routine business was transacted.

March 30th, 1915—4.30 p.m.

The Secretary's report was presented and accounts to the amount of £76, including donation of £50 to the Belgian Relief Fund, were passed for payment.

Nominations as follows were approved:—Members—Messrs. C. E. K. Smith and R. A. Rolfe.

Professor D. B. Waters was appointed to fill the casual vacancy in the office of Vice-President.

A Statement of estimated receipts and expenditure for 1915 was submitted and adopted.

Resolved that the scheme of Incorporation with remarks thereon be printed and submitted to members for suggestions.

Decided that the recommendation to hold the First Ordinary Meeting, 1916, in the South Island of New Zealand be submitted to Councillors for expression of opinion.

Resolved that a report on the subject of Certification of Mining Managers be submitted to the next meeting of the Committee.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Applications for admission to the Institute have been received from the following, whose names are herewith submitted to Members for confidential report:—Members—Messrs. HARTWELL CONDER, Strahan, Tas.; FREDERIC W. R. GODDEN, Boulder, W.A. and LESLIE V. WATERHOUSE, Queenstown, Tas. Associate Members:—Messrs. JOHN K. CAMPBELL, Stannary Hills, Q.; VALENTINE A. CORSTOPHAN, Broken Hill, N.S.W. and BRIAN W. LENNON, Hawthorn, Vic.

Since the publication in Proceedings No. 16 of a supplementary list of Members, the following have been admitted:—

MEMBERS.

NAMES AND ADDRESSES.

BARAGWANATH, WILLIAM, Geological Survey, Ballarat, Vic.
HASSON, ROBERT, Hagelthorne-street, Wonthaggi, Vic.
MURDOCH, FREDERICK M., Pernatty Mines, Port Augusta, S.A.

ASSOCIATE MEMBERS.

ALLEN, HECTOR R., Brazil-street, Broken Hill, N.S.W.
BARBER, CHARLES S., "The Cloister," Peel-st., Broken Hill, N.S.W.
CAMPBELL, ALEXANDER G., "Kia Ora," Bay-rd., Sandringham, Vic.
ELFORD, CHARLES, Three Sisters Mines, *via* Townsville, Q.
GETTY, ANDREW, 7 Victoria-street, Kalgoorlie, W.A.
HARRISON, PERCY H., Great Boulder Propty. Co., Boulder, W.A.
LEWIS, KEITH B., "Remo," 41 Kooyong-road, Armadale, Vic.
MACGRUER, DONALD C., New Ravenswood Ltd., Ravenswood, Q.
POWELL, ARTHUR E., Mount Cuthbert, Q.
STURZAKER, NEVILLE, Mount Cuthbert, Q.
WILLIAMS, ARTHUR, Mount Gordon, *via* Cloncurry, Q.
WINTERS, ROBERT J., Department of Mines, Darwin, N.T.

ALTERATION OF RULES.

At a Special Meeting of Members of the Institute, held on 29th January, the alterations to Rule VII., clauses 2, 6 and 7, recommended by the Council, were confirmed by ballot.

INCORPORATION.

In regard to the matter of Incorporation announced in No. 16 of Proceedings, a recommendation was made at the First Ordinary Meeting, Mt. Lyell, that the scheme as suggested, together with suggestions to date thereon be submitted to Members. The Council has the matter in hand.

BELGIUM RELIEF FUND.

A recommendation to the Council was made at the First Ordinary Meeting, Mt. Lyell, that the sum of £50 be donated to the Belgium Relief Fund. This donation has since been made.

FIRST ORDINARY MEETING, 1916.

The Council was advised at the Mt. Lyell meeting to take into consideration the question of holding the First Ordinary Meeting, 1916, in the South Island of New Zealand. An announcement thereon will be made in due course.

PUBLICATIONS.

Papers appearing in this number are :—

- Mining Education in Australasia. By D. B. Waters.
- North Mine Practice in Underground Points and Crossings. By A. G. Campbell.
- Notes on Lake Margaret Hydro-Electric Power Scheme. By Geo. W. Wright.
- The Application of Surface Combustion. By J. H. Ledeboer.

Discussions are invited thereon.

ON ACTIVE SERVICE

The Council has decided that all members who have been accepted for active service with the Commonwealth Expeditionary Forces be exempted from payment of subscriptions during their period of service.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

FROM 31ST DECEMBER, 1914, TO 31ST MARCH, 1915.

Australian Mining Standard	weekly	Melbourne
Australian Mining and Engineering Review	monthly	Melbourne
Engineering and Mining Journal	weekly	New York
Iron and Coal Trades Review	weekly	London
Mining Journal	weekly	London
Mining Press	weekly	San Francisco
The Colliery Engineer	monthly	Scranton, Pa.
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa :	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Chamber of Mines of Victoria : Monthly Mining Report		Melbourne
Chamber of Mines of Western Australia : Journal	monthly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines : Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Production		Johannesburg
Rhodesia Chamber of Mines : Report of Executive	monthly	Bulawayo
Institution of Mining Engineers : Transactions. Vol. XLVII., Parts 6 and 7; Vol. XLVIII., Parts 1 and 2		London

Institution of Mining and Metallurgy :	
Bulletins, Nos. 121-124 - - -	London
Geologists' Association :	
Proceedings, Vols. XXV., Parts 5, 6 ; VXVI., Part 1	London
North of England Institute of Mining and Mechanical Engineers : Annual Report, 1913-14 - -	Newcastle
Department of Mines, Canada :	
Geological Survey, Memoir, No. 20 E	
The Copper Smelting Industries of Canada	
Lode Mining in Tukon	
Museum Bulletin, Nos. 4 and 8	
Summary Report, 1913	
Report on Building and Ornamental Stones of Canada, Vol. II. - - -	Ottawa
Bureau of Mines, Canada :	
Report, Vol. XXII., Parts I. and II., 1914 -	Toronto
American Institute of Mining Engineers :	
Bulletins, Nos. 93-98 - - -	New York
American Society of Civil Engineers :	
Transactions, Vol. LXXVII., Dec., 1914 -	New York
United States Geological Survey :	
Bulletins, Nos. 548, 550, 556, 557, 571, 574, 579 585	
Water Supply Papers, Nos. 323 ; 327; 581, A, 580, B, D, E ; 345, E, F ; 340, B	
Professional Papers, Nos. 83, 90, C, D -	New York
Colorado Scientific Society :	
Proceedings, Vol. X., pp. 415-452 - -	Denver
Field Museum of Natural History :	
Annual Report of Director, 1913	
Publication, 173 - - - -	Chicago
Geological Survey of India :	
Record, Vol. XLIV., Part 3 - - -	Calcutta
Department of Mines, Victoria :	
Geological Survey	
Bulletin, No. 33, Hustler Line of Reef, Bendigo	
Bulletin, No. 34, Economic Geology and Mineral Resources of Victoria	
Bulletin, No. 35, The New Jubilee Mines, Scarsdale	Melbourne

Royal Society of Victoria :
Proceedings, Vol. XXVII., Part 1 - - Melbourne
Royal Society of New South Wales :
Journal and Proceedings, Vol. XLVII., Part III.
Vol. XLIII., Part I. - Sydney
The Royal Zoological Society of New South Wales :
The Australian Zoologist, Vol. I., Part 2 - Sydney
Department of Mines, South Australia :
Review of Mining Operations in S.A., to June 30th, 1914, No. 20 - - - - Adelaide
Royal Society of South Australia :
Transactions, Vol. XXXVIII. - - Adelaide
Department of Mines, Tasmania :
Geological Survey Bulletin No. 18
Geological Survey Report, No. 4, 5 - - Hobart
Société des Ingénieurs Civils de France :
Memoirs, Series 7, No. 6 - - - Paris
Società Geologica Italiana :
Bulletin, Vol. XXXIII., No. 2 - - Rome

Concentrate Treatment Costs at the Treadwell Cyanide Plant. W. P. Lass. (8) Dec. 19, 1914.
A Rapid Method for Washing Gold Beads. Wm. Spence Black. (6) Dec. 26, 1914.
Tube Milling at Cobalt. John G. Muller. (6) Sept. 26, 1914.
Coal-Dust Reverberatory Firing. E. P. Mathewson. (6) Oct. 24, 1914.
Some of the Difficulties Encountered in Analytical Operations. Gordon Surr. (5) Dec. 26, 1914.
The Mechanical Drying of Ore Products. C. O. Bartlett. (5) Dec. 26, 1914.
Leaching of Copper Ores: Discussion. (5) Dec. 26, 1914.
Precipitation Plant at the Copper Queen Mines. (6) Jan. 2, 1915.
Rectangular Mechanical Furnaces. Gordon Vivian. (6) Jan. 23, 1915.
Chloridizing Blast Furnace Roasting and Leaching. Glenn A. Keep. (6) Serial commenced Feb. 6, 1915.
Laist Roasting Patents. (6) Feb. 6, 1915.
Recent Progress in the Design of Ore-Milling Plants. Colby M. Avery and C. A. Tupper. (5) Jan. 2, 1915.
Progress in Hydro-Metallurgy in 1914. (5) Jan. 2, 1915.
Electrolytic Zinc: the Reed Process. E. H. Leslie. (8) Jan. 2, 1915.
Electric Smelting in 1914. Robert M. Keeney. (8) Jan. 2, 1915.
Danish Tube-Mill Pebbles and Substitutes. (8) Jan. 23, 1915.
Ore Treatment by the Vandercook Process. (8) Feb. 13, 1915.
Coal-Dust-Fired Reverberatory Furnaces of the Canadian Copper Co. David H. Browne. (29) Bull. No. 97.
Coal-Dust-Fired Reverberatories at Washoe Reduction Works. Louis V. Bender. (29) Bull. No. 97.
Experiments on the Flow of Sand and Water through Spigots. R. H. Richards and Boyd Dudley, jun. (29) Bull. No. 97.
Titanium and its Effects on Steel. George F. Comstock. (42) Feb., 1915.

MISCELLANEOUS.

A New Core Remover for Diamond Drilling. (5) Oct., 31, 1914.
Modern American Rock-Drills. L. O. Kellog. (6) Serial commenced Oct. 17, 1914.
Coal-Dust Explosion Investigations. M. J. Taffenal. (29) Bull. No. 93.
Investigations of Coal-Dust Explosions. George S. Rice. (29) Bull. No. 94.
A New Safety Detonating Fuse. Harrison Souder. (29) Bull. No. 94.
The Use of Mud-Laden Water in Drilling Wells. I. N. Knapp. (29) Bull. No. 96.
Men and Machinery of the Comstock: the Combination Shaft. G. W. Dickie. (6) Dec. 5, 1914.
Scientific Purchasing of Mining and Milling Machinery. Glenville A. Collins. (5) Dec. 19, 1914.
Badwin Mines of the Burma Corporation. (6) Jan. 23, 1915.
Using Concrete in Mine and Mill. S. R. Stone. (5) Jan. 2, 1915.
Preservation of Timber in Mining. F. K. R. Moll. (8) Jan. 9, 1915.
The Limits of Mining Under Heavy Wash. Douglas Bunting. (29) Bull. No. 97.
Safeguarding the Use of Mining Machinery. Frank H. Kneeland. (29) Bull. No. 97.
The Testing and Application of Hammer Drills. Benjamin F. Tillson. (29) Bull. No. 98.
Tempering of Machine and Hand Drills. Hugh Spottiswood. (2) March, 1915.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 18.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

1915.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

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The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 SWANSTON STREET, MELBOURNE, VICTORIA.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Abstract).

APRIL 27TH, 1915—4.30 P.M.

The Secretary's report was presented and accounts to the amount of £32, were passed for payment.

Nominations as follows were approved:—Associate—Mr. Bertram Whitington. Subscriber—Mr. R. H. S. Abbott.

A paper by Mr. V. F. Stanley Low, entitled "An Example of Low Working Costs," was referred to the Publication Committee.

It was decided to exchange publications with Silby College, Cornell University, and the Sheffield Scientific School, Yale University.

Other routine business was transacted.

MAY 31ST, 1915—4.30 P.M.

The Secretary's report was presented and accounts to the amount of £119, including printing Proceedings £63, and Rent £24, were passed for payment.

Nomination of Mr. Claude Henry Moxon as a Member was approved.

Resignation of Mr. Will Owen was accepted.

The Michigan College of Mines and the Northern Engineering Institute, N.S.W. were placed on the exchange list.

The report of the Publication Committee, which included a recommendation to publish an Index of all papers published by the Institute up to the end of 1914, was adopted.

It was reported that the Minister of Public Works, Victoria, had been requested to exempt Members of the Australasian Institute of Mining Engineers from the operations of the proposed Architects' Registration Act.

The question of fixing the locality of the First Ordinary Meeting, 1916, was deferred for consideration at a future meeting.

Judges were appointed to examine students' essays submitted for the Dank's prize, 1915.

Other routine business was transacted.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Applications for admission to the Institute have been received from the following, whose names are herewith submitted to Members for confidential report:—Members—Messrs. GEORGE BARKER, Rosebery, Tas.; CYRIL W. GUDGEON, Waratah, Tas.; and CLAUDE HENRY MOXON, Williamsford, Tas. Associate—BERTRAM WHITINGTON, Queenstown, Tas.

PUBLICATIONS.

Papers, &c. appearing in this number are:—

PAPER—An Example of Low Working Costs. By V. F. Stanley Low.

DISCUSSION—Mining Education in Australasia.

An Index of papers published by the Institute from its inception to the end of 1914 is issued as a supplement.

The attention of Members is directed to the fact that the Institute has for sale the following publications, viz.:—"The Geology of Mount Lyell" by J. W. Gregory, D.Sc., F.R.S. and "The Geology of Kalgoorlie" by C. O. G. Larcombe.

LIST OF MEMBERS OF THE INSTITUTE SERVING AT THE FRONT.

AARONS, Capt. J. BOYD, Australian Imperial Forces.
ALEXANDER, HUBERT, Australian Expeditionary Forces.
ANDERSON, E. S., ,, ,, ,,
AVERY, WILFRED T., ,, ,, ,,
BARBER, Lieut CHARLES S., ,, ,, ,,
BELL, WILLOUGHBY G., ,, ,, ,,
BEST, G. H. T., King Edward's Horse.
BRAY, Lieut. FRANCIS P., Royal Engineers.
CAMPBELL, Lieut. A. G, Australian Expeditionary Forces.
CASEY, Lieut. R. G. junr. (Aide-de-camp to General Walker), Australian Epeditionary Forces.

CONNOR, Lieut. J. L., Australian Expeditionary Forces.
FRASER, DONALD L., ,, ,, ,,
FRASER, E. H., ,, ,, ,,
GABRIEL, E. ESCOTT, ,, ,, ,,
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KELLY, M. B. H., Australian Expeditionary Forces.
LEWIS, K. B., ,, ,, ,,
MARKS, DOUGLAS G., ,, ,, ,,
McBRYDE, JAMES, King Edward's Horse.
MOORE, R. INGRAM, Australian Expeditionary Forces.
PEARSON, H. F., ,, ,, ,,
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TOWNSEND, HAROLD, ,, ,, ,,
TUCKER, VIRGIL, ,, ,, ,,
WATT, Lieut. RAYMOND T., Australian Expeditionary Forces (Killed in Action).

SUBSCRIPTIONS.

☞ Members are reminded that Subscriptions for 1915 are now due, and are requested to forward same as early as possible.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

From 31st March, 1915, to 30th June, 1915.

Australian Mining Standard	weekly	Melbourne
Australian Mining and Engineering Review	monthly	Melbourne
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Iron and Coal Trades Review	weekly	London
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Mining Press	weekly	San Francisco
The Colliery Engineer	monthly	Scranton, Pa.
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa :	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Chamber of Mines of Victoria : Monthly Mining Report		Melbourne
Chamber of Mines of Western Australia : Journal	monthly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines : Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Production		Johannesburg
Rhodesia Chamber of Mines : Report of Executive	monthly	Bulawayo
Institution of Mining Engineers : Transactions. Vol. XLVIII., Parts 3 and 4; Vol. XLIX., Part 1		London
Institution of Mining and Metallurgy : Transactions, Vol. XXIII., 1913–14; Bulletins, Nos. 125-127		London

Geologists' Association :
Proceedings, Vol. XXV., Parts 5 and 6
Vol. XXVI., Parts 1 and 2 - London

Royal Dublin Society :
Scientific Proceedings, Vol. XIV., Nos. 17-23
Economic Proceedings, Vol. II., Nos. 8 and 9 - Dublin

Department of Mines, Canada :
Geological Survey, Memoir, Nos. 61 and 65 - Ottawa

Canadian Mining Institute :
Transactions, Vol. XVII., 1914 - - Montreal

American Institute of Mining Engineers :
Transactions, Vols. XLVI., and XLVII.
Bulletins, Nos. 99-101
List of Members, 1915 - - - New York

United States Geological Survey :
Bulletins, Nos. 537, 540, 541, 543, 546, 547, 549, 551-554, 558, 561, 564, 570, 572, 575-578, 583, 584, 586, 590, 592, 599, 600 : 580, A, B, C, F, G, H, I, J, K, M, N, O : 581, C. D.
Thirty-ninth Annual Report 1914
Water Supply Papers, Nos. 321-326, 328-330, 335, 336, 339, 344, 346-348, 363, 364, 309 ; 340, A, B, C, D, E ; 345, A, B, C, D, G
Professional Papers, Nos. 81, 82, 84 ; 85, D ; 90, A, B ; 85, E
Mines Reports, Nos. I, A, 6-12, 14-26 ; II, 14, 17, 18, 20-35
Mineral Resources of the United States, Parts 1 and 2, 1912 - - - New York

State of Missouri Bureau of Geology and Mines :
Biennial Report of the State Geologist
Geology of the Rolla Quadrangle, Vol. XII., 2nd Series - - - - Rolla

California State Mining Bureau :
Petroleum Industry of California, Bulletin 69
,, ,, ,, Maps
Mines and Mineral Resources of Imperial County
,, ,, ,, San Diego County Sacramento

Mysore Geological Department :
Report of the Chief Inspector of Mines, 1913-14 - Bangalore

Geological Survey of India :	
Record, Vol. XLV., Part 1 - - -	Calcutta
Royal Society of Victoria :	
Proceedings, Vol. XXVII., New Series, Part 2 -	Melbourne
Department of Mines, New South Wales :	
Annual Report, 1914 - - -	Sydney
Royal Society of New South Wales :	
Journal and Proceedings, Vol. LVIII., Part II. -	Sydney
Royal Society of Queensland :	
Proceedings, Vol. XXVI.	
Index Proceedings, Vols. I.-XXV. - -	Brisbane
Department of Mines, South Australia :	
Review of Mining Operations in S.A., to Dec. 31st. 1914, No. 21 - - - -	Adelaide
Department of Mines, West Australia :	
Geological Survey, Bulletin Nos. 56, 59 and 62 -	Perth
Royal Society of Tasmania :	
Papers and Proceedings, 1914 - -	Hobart
Queensland Government :	
Report of the Second Interstate Conference on Artesian Water, Brisbane,, 1914 -	Brisbane
Société des Ingénieurs Civils de France :	
Memoirs, Series 8, Nos. 6-12	
Proceedings, 1915, Nos. 1-3 - - -	Paris
Real Academia de Cienceas Exactas Fisicas Naturales :	
Revista, Vol. XII., Nos. 8-12 - - -	Madrid
El Museo Nacional :	
Annales Del Museo Nacional De Historia Natural Tomo XXV. - - - -	Buenos Ayres

RECENT ARTICLES ON MINING MATTERS.

(31st March, 1915, to 30th June, 1915.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly; mth., monthly.

(1) *The Australian Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk. 6d.

(5) *Mining and Engineering World*, Chicago, wk., 10c.

(6) *The Engineering and Mining Journal* New York, wk., 15c.

(7) *The Colliery Engineer*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

(11) *Publications*, Department of Mines, Sydney, New South Wales.

(12) *Publications*, Department of Mines, Adelaide, South Australia.

(13) *Publications*, Department of Mines, Brisbane, Queensland.

(14) *Publications*, Department of Mines, Perth, Western Australia.

(15) *Publications*, Department of Mines, Hobart, Tasmania.

(16) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(17) *Publications*, Bureau of Mines, Toronto, Ontario.

(18) *Publications*, Geological Survey of India, Calcutta.

(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(24) *Quarterly Journal*, Geological Society, London.

(25) *Transactions*, Inst. Mining and Metallurgy, London, E.C.

(26) *Transactions*, Inst. Min. Eng., London.

(27) *Journal*, Canadian Mining Inst., Ottawa Ontario.

(28) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(29) *Transactions*, Am. Inst. of Min. Eng. New York City.

(30) *Proceedings*, Colorado Scientific Soc. Denver, Col.

(31) *Journal*, Franklin Inst., Philadelphia, Pa.

(32) *Australian Mining and Engineering Review*, Melbourne, Vic., mth., 6d.

(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins*, Société des Ingénieurs Civils, Paris.

(35) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(36) *Publications*, Iron and Steel Institute, London.

(37) *Proceedings*, Inst. of Mech. Eng., London.

(38) *Publications*, Field Columbian Museum Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

(41) *Publications*, Department of Mines, Wellington, N.Z.

(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(44) *Proceedings*, Geologists' Association, London.

LIST OF ARTICLES.

ELECTRICAL.

Electric Hoists and Some of the Recent Installations. Girard B. Rosenblatt. (5) Feb. 20, 1915.
Some Results of B., A. and P. Railway Electrification. (6) March 20, 1915.
Hydro-Electric Plant of the Cerro de Pasco Mining Co., Peru. Guillermo Hartmann. (5) April 3, 1915.
Progress in Electric Haulage. (5) April 10, 1915.
The Electrical Driving of Winding Engines and Rolling Mills. (27) Trans., Vol. XVII.
The Electric Furnace in the Foundry. William G. Kranz. (29) Bull. No. 101.
The Rennerfelt Electric Furnace. (6) Feb. 27, 1915.

GEOLOGICAL.

The Search for Ore. Geo. F. Collins. (2) April, 1915.
Notes on the Geology of Meekatharra, Murchison Goldfield, and Surrounding Country. E. de C. Clarke. (42) April, 1915.
On Carnolite Deposits and the Rand Banket. Chester T. Kennan. (8) April 17, 1915.
The Application of the Apex Law at Wardner. Fred. T. Greene. (29) Bull. No. 101.
Method of Making Mineralogical Analysis of Sand. C. W. Tomlinson. (29) Bull. No. 101.
Geology of the Burro Mountains Copper District, New Mexico. R. E. Somers. (29) Bull. No. 101.

MECHANICAL.

Acetylene Welding of Pipe Joints. George H. Manlove. (5) March 20, 1915.
Noteworthy Progress of the Diesel Engine. E. A. Garratt. (5) March 20, 1915.
Automatic Skip Hoist at Morenci Mill. H. L. Hall. (5) April 10, 1915.
Distribution of Heat Energy and Frictional Losses in Internal-Combustion Engines. Prof. John Eustice. (37) March, 1915.
Convertible Combustion Engines. Alan E. L. Chorlton. (37) March, 1915.
A Graphical Method of Finding Inertia Forces. (37) No. 4, 1915.

METALLURGICAL.

Assay of Precious Metal Bullion. (6) Feb. 20, 1915.
The Cost of Cyaniding. Herbert A. Megraw. (6) Feb. 13, 1915.
Rapid Analysis of Alloys for Tin, Antimony, and Arsenic. F. A. Stief. (43) March, 1915.
The Theory of Tube-Milling. H. A. White. (28) Feb., 1915.
Electro-magnetic Ore Separation. I. C. Clark. (6) Mar. 20, 1915.
Magnetic Concentration Mill at Mount Hope. Samuel Sharpia. (6) March 27, 1915.
Aluminium Precipitation at Nipissing. E. M. Hamilton. (6) March 27, 1915.
Zinc Pigments and the Leadville Works. E. H. Leslie. (8) March 20, 1915.
A Simple Sieve Shaker. W. S. Black. (8) March 20, 1915.
Platinum Analysis. F. Mylius and A. Mazzucchelli. (8) March 27, 1915.
The Smelting of Dross in the Electric Furnace. Raymond S. Wile. (5) March 13, 1915.
Knight-Christensen Process at Provo, Utah. Chas. F. Spaulding. (5) March 20, 1915.
Some of the Problems in Copper Leaching. L. D. Ricketts. (5) April 10, 1915.
Progress in Metallurgy. James Douglass. (8) April 10, 1915.
The Mechanical Efficiency of Crushing. Herman C. Kenny. (8) April 10, 1915.
Smoke Dilution at Midvale. L. S. Austin. (8) April 10, 1915.
Cyaniding of Gold-Silver Ores at Waihi Grand Junction. Noel Carless. (25) Bull. No. 127.
The Effect of Mineralized Waters in Cyanide Plants. Thos. B. Stevens and W. S. Bradley. (25) Bull No. 127.
Treatment of Arsenical–Antimonial Sulphide Ores at Youanme. K. Byron Moore and H. R. Edmands. (42) Feb., 1915.
Leaching with Ammonia. J. D. Audley Smith. (35) Feb., 1915.
Some Problems in Copper Leaching: A Discussion. (29) Bull. No. 100.

Pyritic Smelting: A Discussion. (29) Bull. No. 100.
Turbo Blowers for Blast Furnace Blowing: A Discussion. (29) Bull. No. 100.
Pyritic Smelting. Geo. A. Guess. (27) Trans., Vol. XVII.
The Commercial Aspect of Electric Zinc-Lead Smelting. Woolsey M'Alpine Johnson. (27) Trans., Vol. XVII.
Extracting Copper with Ammonia. C. H. Bendict. (8) April 17, 1915.
Choosing Rock-Drill Equipment. P. B. M'Donald. (8) April 17, 1915.
Leaching of Copper Ores by the Hoffman Process. Frederick J. Pope. (8) April 24, 1915.
The Flotation of Copper Ores. "An Occasional Contributor." (8) May 1, 1915.
Qualitative Blow-Pipe Analysis in the Field. George Delius. (8) May 8, 1915.
Simple and Efficient Precipitant for Copper from Acid Solutions. William Thompson. (5) May 1, 1915.
Tendency of American Milling Machinery Practice. Julius I. Wile. (6) April 17, 1915.
Determination of Platinum, Palladium, and Gold. A. M. Smoot. (6) April 17, 1915.
New Anaconda Leaching and Acid Plants. E. P. Mathewson. (6) April 24, 1915.
Recent Rand Metallurgical Practice. M. Thornton Murray. (6) May 1, 1915.
Notes on the Practical Testing of Working Cyanide Solutions. Edward H. Croghan. (28) April, 1915.
The Use of Scoop Discharges in Tube Mills. Walford R. Dowling. (28) March, 1915.
Refining Gold Bullion: Presidential Address. Sir Thomas Kirke Rose. (26) Bull. No. 127.
The Precipitating Action of Carbon in Contact with Auriferous Cyanide Solutions. W. R. Feldtmann. (26) Bull. No. 127.
The Effect of Mineralized Waters in Cyanide Plants. Thos. B. Stevens and W. S. Bradley. (26) Bull. No. 127.
Metallurgical Practice in the Witwatersrand District, South Africa. F. L. Bosqui. (29) Bull. No. 101.
Fusibility of Coal Ash in Various Atmospheres. A. C. Fieldner and A. E. Hall. (43) May, 1915.
Practical Methods for the Determination of Radium. S. C. Lind. (43) May, 1915.
Practical Points on Sampling. Arthur Feust. (8) March 6, 1915.

MISCELLANEOUS.

Design of Angle-Sheave Frames. Floyd L. Burr. (6) Series commenced Feb. 20, 1915.
Oxyacetylene Welding in Mining. (6) Feb. 27, 1915.
Reinforced Concrete Shaft Sets. L. D. Davenport. (6) March 6, 1915.
Misfires in Blasting and Their Causes. J. B. Burgess. (8) Feb. 27, 1915.
Caving System at the Ohio Copper Mine. F. Sommer Schmidt. (8) March 6, 1915.
Visualizing Working Conditions in a Mine. H. N. Stronck and J. R. Billyard. (8) March 20, 1915.
Economical Use of Explosives. R. Noblett. (8) March 20, 1915.
If We Had Known Then What We Know Now. C. F. Spaulding. (5) March 27, 1915.
Some Uses of Traverse Tables. James Underhill. (8) April 3, 1915.
Advantages of a Level Stope. H. Y. Russell. (2) May, 1915.
Methods for Preservation of Mine Timbers. G. B. M'Donald. (2) May, 1915.
Latent Errors in Mine Sampling. Morton Webber. (2) May, 1915.
The Cementation Process for Sinking Shafts. Sydney F. Walker. (2) May, 1915; also (35) Feb., 1915.
The Training of Mining Engineers. R. W. Dron. (26) Vol. XLIX., Part 1.
The Nature of Explosives. Prof. Harold B. Dixon. (26) Vol. XLIX., Part 1.
Power Costs. William B. Woodhouse. (26) Vol. XLIX., Part 1.
The Valuation of Mines. J. D. Kendall. (27) Trans., Vol. XVII.
Notes on Mine Sampling. Victor G. Hill. (27) Trans., Vol. XVII.
The Human Side of Mining. Jas. F. Kemp. (8) April 24, 1915.
Knowledge and Research. R. W. Raymond. (8) May 1, 1915.
Cost and Efficiency. Heath Steele. (8) May 8, 1915.
Labour-Saving Devices. S. A. Worcester. (8) May 8, 1915.
Gold Dredging in the Philippines. Wm. Kane. (6) April 17, 1915.
Safety Measures in the Use of Explosives. Wm. Cullen. (28) March, 1915.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 19.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

1915.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

LIST OF OFFICERS, 1915.

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D. B. WATERS, Dunedin, N.Z.

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R. S. BLACK	Western Australia
H. W. GEPP	New South Wales
RICHARD HAMILTON	Western Australia
STANLEY B. HUNTER	Victoria
A. S. KENYON	Victoria
A. H. MERRIN	Victoria
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JOHN W. MOULE	Queensland

Secretary:

D. L. STIRLING, Melbourne, Vic.

LOCAL CORRESPONDENTS:

South Australia	JAMES P. WOOD, Adelaide.
New South Wales	F. DANVERS POWER, Sydney.
New Zealand	D. B. WATERS, Dunedin.
Tasmania	RUSSELL M. MURRAY, Mount Lyell.
North Queensland	MURRAY RUSSELL, Cloncurry.

BRANCH SECRETARIES:

Kalgoorlie	F. W. R. GODDEN, Ivanhoe Gold Corporation, Boulder, W.A.
Broken Hill	J. M. BRIDGE, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Mount Morgan	B. G. PATTERSON, Mount Morgan, Q.

The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 SWANSTON STREET, MELBOURNE, VICTORIA.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Abstract).

JUNE 28TH, 1915—4.30 P.M.

The Secretary's report was presented and accounts to the amount of £25 were passed for payment.

Nomination of Mr. George Barker as a Member was approved.

The following were admitted as Students:—Messrs. Hubert Henry Carroll and Walter Gilbert Langford.

A suggestion that the Institute secure representation on Faculties and Boards controlling Mining Education was favorably considered and referred to a Sub-Committee for report.

Consideration was given to the question of the limitation in numbers of visitors accompanying members on the annual visits to mining centres. It was resolved that the names of members and visitors wishing to attend such meetings should reach the Secretary at least a fortnight prior to the meeting.

A suggestion to hold the First Ordinary Meeting, 1916, in the South Island of New Zealand was further discussed and a

recommendation submitted that members should be requested to notify the Secretary as to the probability of their attending the meeting, if decided upon.

Resolved that members attending meetings be supplied with a small badge.

JULY 26TH, 1915—4.30 P.M.

The Secretary's report was presented and accounts to the amount of £84, including Broken Hill Branch £50, were passed for payment.

Nominations as follows were approved:—Members—Messrs. Cryil W. Gudgeon and Stanley R. Mitchell.

Resignation of Mr. R. C. Inglis was accepted.

The question of holding the First Ordinary Meeting, 1916, in the South Island of New Zealand was further discussed. The general feeling of the meeting was that, owing to the abnormal conditions, on account of the war, it would be inadvisable to hold a meeting next year.

Further consideration was given to the question of visitors attending meetings and the matter was deferred for further consideration until the next meeting of the Executive Committee.

Resolved that members be notified that they may, if they so desired, have their four quarterly Proceedings bound at the end of the year in one volume at the cost of 5s. 6d., plus postage 1s. If new copies were desired, the cost to the Institute of same would be charged.

AUGUST 30TH, 1915—1 P.M.

The Secretary's report was presented and accounts to the amount of £110, including printing Proceedings and Index, £82, were passed for payment.

Resolved that £250 worth of Commonwealth War Loan be purchased.

The following were elected to the Institute:—As Members—Messrs. George Barker, Hartwell Conder, F. W. R. Godden (transfer from Associate Member), Cyril W. Gudgeon, Brian W.

Lennon, Claude H. Moxon, Richard A. Rolfe, Claude E. Kennedy Smith and Leslie V. Waterhouse (transfer from Associate Member). As Associate Members—Messrs. Arthur E. Tandy, J. K. Campbell, V. A. Corstophan and Bertram Whitington. As Student—Mr. Harry Vernon Pethebridge.

At the invitation of the State Munitions Committee, it was resolved that the Institute become a Sub-Committee thereof.

The following papers were submitted and referred to the Publication Committee :—"Mining Methods at Mount Lyell," by R. M. Murray; "Stope-Survey Practice at Mount Lyell," by G. F. Jakins and L. J. Coulter; "Lake Margaret Hydro-Electric Power Scheme, Mount Lyell, Tas.," by Geo. W. Wright.

It was resolved that First Ordinary Meetings of the Institute be confined to persons directly interested in mining and allied sciences and that the proceedings be restricted to technical and mining business only.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

The application of Mr. Stanley R. Mitchell, of Melbourne, for transfer from the grade of Associate Member to Member was approved.

Since the publication in Proceedings No. 17 of a supplementary list of members, the following have been admitted :—

MEMBERS.

Names and Addresses.

Barker, George, Rosebery, Tas.
Conder, Hartwell, Strahan, Tas.
Godden, Fred. W. R., Boulder, W.A. (Transfer from Associate Member).
Gudgeon, Cyril W., Waratah, Tas.
Lennon, Brian W., 7 Cristobel Crescent, Hawthorn, Vic.
Moxon, Claude H., Williamsford, Tas.
Rolfe, Richard A., Irvinebank, North Q.
Smith, Claude E. Kennedy, Mulline, W.A.
Waterhouse, Leslie V., Queenstown, Tas. (Transfer from Associate Member).

ASSOCIATE MEMBERS.

Campbell, John K., Stannary Hills, Q.
Corstophan, Valentine A., Mount Jasper, Tas.
Tandy, Arthur E., Mt. Oxide, *via* Cloncurry, North Q.
Whitington, Bertram, School of Mines, Queenstown, Tas.

STUDENTS.

Carroll, Hubert Henry, Mount Lyell, M. & R. Co., Gormanston, Tas.
Langford, Walter Gilbert, Engineering School, The University, Melbourne, Vic.
Pethebridge, Harry V., 48 Fellows Street, Kew, Vic.

BROKEN HILL BRANCH.

Mr. J. M. Bridge has been elected Hon. Secretary of the Broken Hill Branch.

PUBLICATIONS.

Papers, etc. appearing in this number are :—

PAPERS—Presidential Address—Pyrite Smelting at Mount Lyell. By Robert C. Sticht.

Mining Methods at Mount Lyell. By R. M. Murray.

Stope-Survey Practice at Mount Lyell. By G. F. Jakins and L. J. Coulter.

Lake Margaret Hydro-Electric Power Scheme, Mount Lyell, Tas. By Geo. W. Wright.

Notes on the Treatment of Stannite Ore at Zeehan, Tas. By J. H. Levings.

DISCUSSION—Mining Education in Australasia (D. B. Waters). By Wm. Pooie.

BINDING PROCEEDINGS.

Members are hereby notified that they may, if they so desire, have their four quarterly Proceedings bound in dark cloth, lettered in gold on back, at a cost of 5s. 6d., plus cost of postage, 1s. If new copies are desired the bare cost to the Institute of same will be charged, but numbers sent out quarterly may be returned and bound instead. The price of new copies is :—1912, 5s. 6d.; 1913, 7s. 6d.; 1914, 10s. 9d.

FIRST ORDINARY MEETING, 1916.

Owing to the abnormal conditions existing on account of the war, the Executive Committee is of opinion that it would be inadvisable to hold the usual First Ordinary Meeting in 1916.

ANNUAL MEETING, 1916.

The Annual Meeting, 1916, will be held at the rooms, 57-59 Swanston-street, Melbourne, on Friday, 28th January, 1916, at 12 o'clock (noon). The business of the meeting will include the presentation of the Annual Report and Balance Sheet for 1915, and the election of officers. The Officers retiring are :— President—Mr. Robt. C. Sticht; Vice-President—Mr. H. C. Bellinger; Councillors—Messrs. G. D. Delprat, R. S. Black, H. W. Gepp, R. Hamilton, S. B. Hunter, A. S. Kenyon and A. H. Merrin. Nominations to fill the vacancies thus occuring should reach the Secretary forty days before the Annual Meeting.

VISITORS AT MEETINGS.

The following resolution by the Executive Committee has been forwarded to members of Council for an expression of opinion :—

> "That the First Ordinary Meeting of the Institute be confined to persons directly interested in mining and allied sciences, and that the proceedings be restricted to technical and mining business only."

A. T. DANKS' PRIZE, 1915.

Mr. W. G. Langford of the Melbourne University has been awarded the "A. T. Danks' Prize, 1915," for the best paper by a Student descriptive of the visit to Mount Lyell, Zeehan and Waratah in March, 1915.

INCORPORATION.

A copy of the Memorandum and Articles of Association as suggested by the Broken Hill Branch, together with comments thereon is published in this number and members are invited to forward expressions of opinion on the proposal.

SUPPLEMENTARY LIST OF MEMBERS OF THE INSTITUTE SERVING AT THE FRONT.

Brown, H. Wheeler,	Australian	Expeditionary	Forces.
Dempster, Geo. C.,	,,	,,	,,
Goode, Kenneth B.,	,,	,,	,,
Grut, Lieut. L. de J.,	,,	,,	,,
Holder, Evan M.,	,,	,,	,,
Horsley, R. D.,	,,	,,	,,
Irvine, Claude L.,	,,	,,	,,
Jenkins, Lieut. Robt. S.,	,,	,,	,,
Key, John F., Imperial Forces.			
Mickle, Kenneth A.,			
McBride, William J.,	Australian	Expeditionary	Forces.
Roberts, Reginald A. J.,	,,	,,	,,
Smith, Eugene,	,,	,,	,,
Southon, Ronald D.,	,,	,,	,,

Killed in Action.

Bayly, Colin, Killed in Flanders.
Connor, Lieut. J. L., Killed in Galipolli.

AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

57 Swanston-street,

Melbourne, 30th September, 1915.

Incorporation.

To Members.

As recommended by members at the First Ordinary Meeting, 1915, the Council presents herewith the scheme of Incorporation as prepared by the Broken Hill branch. Opinions thereon have been obtained from individual Members of Council in the various States, and these opinions are in favour of Incorporation, provided that the difficulty arising through the absence of a Federal Companies Act can be overcome.

The documents published herewith are :—

(1) Proposed Memorandum and Articles of Association as prepared by the Broken Hill branch.

(2) Explanatory notes by the Sub-Committee of the Broken Hill branch.

(3) Memorandum by the Executive Committee.

(4) Opinions of Members of Council (summarized).

These are now submitted to members for expression of opinion before further steps are taken by the Council.

By direction,

D. L. STIRLING,

Secretary.

AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

(Incorporated under the *Companies Act* on the day of .)

Memorandum of Association.

The name of the Association is the Australasian Institute of Mining Engineers and it is hereinafter referred to as the Institute.

1. The term "metals" hereinafter referred to denotes all metals and minerals other than coal.

2. The Registered Office of the Institute will be situate in Australia.

3. The objects for which the Institute is established are—

(*a*) To acquire and take over the whole or any of the assets and liabilities of the unincorporated Australasian Institute of Mining Engineers established.

(*b*) To promote the advancement of Mining Engineering in all or any of its branches, and all matters connected with or relating to the production of metals.

(*c*) To promote and facilitate the interchange of information ideas and practice on all matters pertaining to the profession of Mining Engineering and the consideration and discussion of all questions relating thereto or affecting the same.

(*d*) To invite communications written or oral relating to the production of metals, the finance management and administration of all branches of mining and metallurgy, and to receive hear publish and discuss such communications.

(*e*) To give facilities to and assist the Legislature, Governors of mining schools, public bodies and others in conferring with and ascertaining the views of persons or bodies engaged or interested in the production of metals and all related matters, and to confer with any person or any public body or institution in regard to such matters.

(*f*) To originate and promote improvements in the law, to support or oppose alterations therein, to originate and promote improvements in administration, and for the purposes aforesaid to petition Parliament, promote or oppose Bills, and take such other steps and proceedings as may be deemed expedient.

(*g*) To promote just and honourable practice in the profession of Mining Engineering management and administration, and to suppress malpractice.

(*h*) To watch over, promote and protect the mutual interests of the Members and to provide facilities for promoting and maintaining professional intercourse among them.

(*j*) To provide legal or other professional assistance for the protection of the interests of any Member or Members of the profession in cases which may be deemed to involve questions of principle affecting the profession generally and not the individual interests of the parties litigating only.

Provided that the Institute shall not commit any breach or infringement of the law relating to maintenance or champerty.

(*k*) To establish district, branch or sectional Societies and local and other Associations for the promotion of all or any of the objects of the Institute.

(l) To admit any persons to be Students of the Institute on such terms and to confer on them such rights and privileges as may seem expedient.

(m) To promote and obtain an Act or Acts of Parliament stipulating that the prospectus of every public company floated or offered for flotation in Australasia for the purpose of engaging in any mining enterprise other than the mining of coal shall bear an opinion, description or report on the enterprise signed by a person who shall be registered as hereinafter set forth by the Institute as a qualified Mining Engineer or Metallurgist, but who shall not necessarily be a member of the Institute.

Further, to urge upon Parliament the desirability of appointing a member or adviser to any Royal Commission, Committee, or Court of Enquiry which may be appointed in the Commonwealth or in any of the Australian States or in the Dominion of New Zealand for the purpose of enquiring into or reporting on any matter pertaining to the Mining Industry other than coal, who shall be registered by the Institute as a qualified Mining Engineer or Metallurgist.

(n) To keep a register of Mining Engineers and Metallurgists duly qualified to act as such in Australasia, such register to include all full Members of the Institute and such other persons as the Institute or any body to which it may delegate its duties in this respect may judge to be fully qualified.

4. The income and property of the Institute whencesoever derived shall be applied solely towards the promotion of the objects of the Institute as set forth in this Memorandum of Association, and no portion thereof shall be paid or transferred directly or indirectly by way of dividend, bonus, or otherwise howsoever by way of profit to the Members of the Institute, provided that nothing herein contained shall prevent the payment in good faith of remuneration to any officers, servants, or agents of the Institute or to any Member of the Institute or other person in return for any services actually rendered or to be rendered to the Institute, or prevent the carrying out or giving effect to any of the objects comprised in or referred to in Clause 3 of this Memorandum, or the payment of railway or other fares to Members or persons attending Council or Committee meetings, or the payment in good faith of the out-of-pocket expenses of delegates appointed by the Council to attend the meetings of other Institutions or bodies as representatives of the Institute.

5. The liability of the full Members other than the liability of the amount of the annual subscription is limited to £25 with respect to any action with any outside body in which the Institute may become involved in carrying out this Memorandum of Association.

6. Every Member of the Institute undertakes to contribute to the assets of the Institute in the event of the same being wound up during the time he is a Member or within one year afterwards for payment of the debts and liabilities of the Institute contracted before the time at which he ceases to be a Member, and of the costs, charges and expenses of winding up the same and for the adjustment of the rights of the contributories amongst themselves such amount as may be required, not exceeding £1.

7. If upon winding up, etc., any surplus remains, it shall not be distributed amongst Members, but shall go to found prizes at Universities.

8. True accounts shall be kept, etc.

We, the several persons subscribed, etc.

Articles of Association.

1. In these presents, unless there be something in the subject of context inconsistent therewith, "The Institute" shall mean the Australasian Institute of Mining Engineers.

"Metals" denotes all metals and minerals other than coal.

CONSTITUTION.

2. For the purposes of registration, the number of Members of the Institute is to be taken as 500, but the Council may from time to time register an increase of Members.

3. The Institute shall consist of the following classes of Members, that is to say—

(*a*) Members.

(*b*) Junior Members.

The Institute may have attached to it Students, but a Student shall not be deemed a Member within the meaning of these Articles.

4. The subscribers to the Memorandum of Association and such other persons, being Members and Associate Members of the present constituted Institute, or such other persons as may be eligible in accordance with these regulations, and none other, may be admitted Members or Junior Members of the Institute.

5. Such Members or Associate Members of the present constituted Institute as shall sign and forward to the Secretary so as to be received by him before the day of or in the case of those resident outside Australia before the day of a claim to be entered on the roll of membership according to the form prescribed by the Council, shall, on payment of arrears, if any, due to the said Institute become respectively Members or Junior Members of the Institute without election or payment of entrance fee, provided nevertheless that the "Executive Committee" shall have full discretion to accept any claim received after such respective dates.

QUALIFICATIONS.

6. Subject as aforesaid, the qualifications shall be as follows:—

(I.) *For a Member* it shall be necessary—

(*a*) That he is a mining, engineering, or science graduate of an Australian University or of a University recognized by Australian Universities, or is an Associate of the Ballarat School of Mines, the South Australian School of Mines, the Otago School of Mines, or of any such mining schools of equal standing as may be decided by the Council.

Experience.—And that in addition he has held for a period of four years such a position or positions as shall in the opinion of the Executive Committee have given him sufficient opportunity to acquire knowledge of the control, management or appraisal of mining or metallurgical enterprises.

(*b*) *Practical.*—That he has held for a period of seven years one or any of the following positions:—

(1*b*) Manager or Superintendent of a mine or metallurgical works employing over 20 men.

(2*b*) Chief Engineer, Underground Manager, Chief Metallurgist, Chief Surveyor, Chief Assayer, Assistant Manager of a mine or metallurgical works employing 100 men or over.

(3*b*) Consulting or Reporting Mining Engineer, provided that he shall within the period have examined and *bona fide* reported on at least 30 properties.

(4*b*) Professor or Lecturer in mining, economic geology, industrial chemistry, or metallurgy at a reputable University or mining school.

(*c*) Provided that if any person writes and presents to the Institute any original paper which the Publication Committee of the Council deems to be of sufficient merit. the Executive Committee may remit two years from the practical experience required from that person.

(*d*) That he shall be at least 30 years of age.

(*e*) Or that he is a member of some kindred Association, the requirements for membership of which are in the opinion of the full Council equivalent to those set forth above.

(II.) *For a Junior Member* it shall be necessary—

(*a*) That he is a graduate, etc. (as for Member), and, in addition, has had one year's experience in the practice of mining or metallurgy; or

(*b*) that he has held for a period of four years a position or positions of trust in the practice of mining or metallurgy.

(c) That he is not less than 22 years of age.

(III.) *For a Student.*—May be any person connected with or interested in the mining industry.

ELECTION AND ADMISSION.

7. Every application for admission as a Member or Junior Member, except applications under Article 5, must be made according to the form for the time being required by the Council. This form must be signed by at least four Members or Juniors, of whom two must be full Members, who must from personal knowledge be in a position to and shall vouch for his qualifications. The questions set out in such form must be answered by the candidate to the satisfaction of the Executive Committee, who shall seek the advice of their respective branches where possible in the matter of election, and the candidate must sign the undertaking also therein set out. The form must be forwarded to the Secretary, to be by him submitted to the Executive Committee and approved by them. On election the candidate shall be duly informed thereof by letter, and on payment of his entrance fee and first subscription shall receive a Certificate of Membership of the class to which he has been elected.

8. *Privilege of Membership.*—All Members, Junior Members, and Students shall be equally entitled to the privileges of Membership, provided that Members only shall be entitled to claim registration as qualified Mining Engineers or Metallurgists as set forth in the Memorandum of Association, and provided that Students shall not be entitled to vote, and provided that Junior Members shall not be entitled to vote on any question affecting the constitution of the Institute or its relations with outside bodies.

9. *Transfer.*—It shall be competent for any Junior Member or Student to apply to be transferred to the class of Member or Junior Member, and such application shall be in the form for the time being required by the Council, etc.

10. *Certificates.*—All certificates issued in accordance with the foregoing shall remain the property of the Institute, and shall be returned on demand. The Council may waive the return of any certificate on being satisfied that the same cannot be returned owing to loss, destruction, or other sufficient reason.

11. *Resignation.*—Any Member or Junior Member desiring to resign shall give notice in writing to the Secretary, who shall, as soon as possible, submit it to the Executive Committee, who may accept such resignation. The Executive Committee shall be entitled to require the payment of all subscriptions due to the Institute from, and the return of any certificate issued to, any person so desiring to resign before accepting such resignation.

12. Present rule A.I.M.E., conduct of Members and expulsion.

13. *Subscriptions, Fees, etc.*—Until otherwise determined by the Council and sanctioned by the Members and Junior Members entitled to vote and voting at any General Meeting, and subject as aforesaid, the following fees and subscriptions shall be payable:—

(*a*) Every Member shall, on election, pay an entrance fee of Three Guineas and an Annual Subscription of Three Guineas.

(*b*) Every Junior Member likewise shall pay an entrance fee of Two Guineas and an Annual Subscription of Two Guineas.

(*c*) Every Student shall pay an Annual Subscription of One Guinea for a period covering his first six years, thence onwards an Annual Subscription of Two Guineas.

(*d*) On any transfer from Junior Member to Member made in pursuance of Article 9, a fee of One Guinea shall be paid by the person so transferred.

14. All annual subscriptions shall be due and payable in advance on the First day of January in each year.

15. Every person at whatever time of the year he becomes a Member, Junior Member, or Student, shall forthwith pay his annual subscription as such or the whole of the current year, provided that the first annual subscription so payable by a person becoming a Member, or Junior Member (whether by election or transfer), or becoming a Student after the first day of November shall cover the annual subscription due on the first day of January next ensuing.

16. The election or transfer of any Member, Junior Member, or Student, whose entrance or transfer fee and first subscription is not paid within six months of the date of such election or transfer shall be void.

17. Any Member, Junior Member, or Student whose entrance or transfer fee has not been paid, or whose subscription is more than twelve months in arrears, shall not be entitled to any of the privileges of membership.

18. Any Member or Junior Member whose subscription is more than two years in arrears may, by resolution of the Executive Committee, be struck off the register of the Institute, but shall be liable for the amount due to the Institute at the date of such resolution, provided that any such person so removed may, upon application to the Executive Committee and by payment of such arrears of subscription, be reinstated by the Executive Committee.

19. Every Member or Junior Member shall be liable for the payment of his annual subscription until his resignation has been accepted by the Executive Committee as hereinbefore provided, or until he has forfeited his right to remain in or to be attached to the Institute, and any fee or subscription payable under these regulations shall be a debt due from him to the Institute.

BRANCHES.

20. The Institute may form branches in the various States and in New Zealand, and subdivide them into district divisions, where there is a sufficient number of Members or Junior Members resident in any district. In addition, the Executive Committee, with the sanction of the Council, may at their discretion, upon the receipt of a request from a sufficient number of Members or Junior Members resident in any district, create a district division of the Institute in such district, and they shall have power to dissolve such division at any time after it has been formed.

21. Each Branch shall be constituted and its affairs shall be carried on in accordance with the rules and regulations to be laid down from time to time by the Council. Provided that such rules and regulations shall not be inconsistent with or repugnant to the Memorandum and Articles of Association of the Institute, and shall be submitted to the Registrar of Companies, and shall not be effective until his approval has been obtained.

22. Each division shall be entitled to elect a representative, who may attend the Annual Meeting of the Branch.

23. The place of the Annual Meeting of the Branch shall be decided by a vote of the divisions, each according to its membership.

24. Each Branch, according to its membership, shall elect a Member of Members of Council, who shall, if possible, attend the Annual Meeting or the Council. The Branch will defray the travelling and maintenance expenses of its representatives.

25. The Council will meet at such place as may be determined by the Members voting in General Meeting.

The Council.

Explanatory Clause.—The Council will be the supreme governing body of the Institute. All significant matters will be placed before them individually by letter to obtain their sanction. They have the power of referring any matter back to the Annual General Meeting for further review.

The Council shall be constituted from among the leaders of the profession in the mining world in Australia and New Zealand, and the leaders of the technical branch of the science.

It shall be their province to govern rather than administer.

26. The Council is the supreme governing body of the Institute.

27. The Council shall consist of a President, two Vice-Presidents, Past Presidents of the Institute, and the representatives elected by the branches, together with representatives who may be appointed by the Senate of each University, and the Ballarat, Adelaide and Otago Schools of Mines.

28. The Executive Committee constitutes the administrative body of the Institute (appointed as at present).

29. In the event of a representative of a Branch being unable or unwilling to attend an Annual Council Board Meeting, the said Branch shall have power to select a representative from the Members of the Institute to act at the said Meeting in his stead.

30. *Councillors: Period of Office.*—(*a*) Councillors will be elected for a period of three years, and shall retire in rotation.

31. Each Branch will have the privilege of electing one Member to Council for every 30 financial Members on the roll of the Branch.

32. *Honorary Members of Council.*—The Senate of each Australian University and of each New Zealand University, the Ballarat, Adelaide, and Otago Schools of Mines Councils, and such other technical institutions approved by the Council, shall have in Annual Meeting the privilege of nominating to Honorary Membership of Council one representative each, these representatives to attend the Annual Meeting of the Council.

33. *First Council.*—The subscribers to the Memorandum of Association shall constitute the first Council of the Institute.

34. *Additions to First Council.*—At the end of the first year the Branches will elect their representatives.

The Members of the First Council will hold office for the first four years' period of the Institute.

35. *Privileges of Honorary Members of Council.*—Honorary Members of Council shall have the privilege of voting at General Council Meetings.

They shall each and severally be indemnified against any pecuniary disadvantages arising from their connection with the Institute, except those relating to their travelling expenses.

[Indemnify Honorary Members against any action given against the Institute.]

36. The Council may—

(*a*) Acquire all the property and assets of the present A.I.M.E.

(*b*) Pay the costs, charges and expenses preliminary to and incidental to the formation, establishment and registration of the Institute.

(*c*) Institute, conduct, defend, compound or abandon any legal proceedings by or against the Institute or the Council or the Officers or otherwise concerning the affairs of the Institute, and compound and allow time for payment or satisfaction of any debts due to or any claims or demands by or against the Institute.

(*d*) Make and give receipts, releases and other discharges for money payable to the Institute and for the claims or demands of the Institute.

(*e*) Determine who shall be entitled to sign on behalf of the Institute bills, notes, receipts, acceptances, endorsements, cheques, releases, contracts and other documents; appoint standing

committees for such purposes, delegate to such committees such of their powers as they think fit, and withdraw such powers.

(f) Accept books, money, etc., on behalf of the Institute.

(g) Appoint the Secretary and fix his remuneration.

(h) The Executive Committee may publish such works as they think fit.

37. Subject to the Regulations for the time being, Branches and Divisions shall be so constituted, managed and governed and shall have such powers, rights and privileges, and shall be under such obligations, as shall be prescribed by the Council.

[The Council may appoint as an Executive Committee, &c., as at present constituted.]

Finance.—Advertisements should pay for the production of the journal, or very nearly do so; this will depend entirely upon the Secretary.

To Subs.—£3 3s. each, 300 Members	£945
£2 2s. each, 200 Juniors	420
	£1365

By Travelling Expenses—Councillors	£200
.. ,, ,, Representatives of divisions ..	80
.. Secretary's Salary, say, per annum	500
	£780

Explanatory Notes by the Sub-Committee of the Broken Hill Branch.

A suggested plan to fix the status of the Mining Engineer, and a means of coping with the difficulties mentioned in our Presidential Address of 1914.

To bring our profession into line with the current trend of opinion of Mining Engineers in the older world, and actuated by the remarks of our President in his Presidential Address of 1914, your Broken Hill branch of the Institute appointed a sub-committee, consisting of Messrs. Barton, Bridge, Donaldson, and Clayton to go into the matter and endeavour to suggest some means of bringing our profession into line with the older-established professions and to effectively cope with various disabilities that we are labouring under at the present time.

In Mr. Herman's Presidential Address of 1914 he made certain remarks ......(see p. 80 and p. 87): and again, Mr. Watt, then Premier of Victoria ......(see p. 100). Your Sub-Committee considered it essential that there should be a governing body which would draw all members of the pro-

fession in Australia and New Zealand into closer union—a body which would effectively carry on the administration of the whole profession.

The Australasian Institute of Mining Engineers, as at present constituted, has as its objects the promotion of the arts and sciences connected with mining, and the welfare of its Members, by means of meetings for the reading and discussion of papers and publishing the same, and it was felt that the Institute should have a greater scope and assume administrative in addition to its present technical functions.

In order to carry out its administrative projects your Institute would need to be more firmly constituted than at present; so, following established world-precedent, it was considered advisable to register under a Companies Act, in order to acquire legal status.

This is the one great difficulty: the Companies Acts are all administered by individual States, and any Act of Parliament dealing with the profession would have to be enacted in each State; otherwise, if one State refused to pass the Act, spurious companies would then register in that particular State, to the detriment of the others. This difficulty was not considered insuperable, as it was thought that by the time we are ready to apply for registration some means will have been found to enable us to approach the Federal Government and that of the Dominion of New Zealand for our necessary constitution.

Associations not for profit are registered under all present Acts, and are accorded special privileges (see New South Wales Companies Acts.)

Expenses incidental to registration are very moderate.

Any application for registration must be accompanied by a Memorandum of Association, together with Articles of Association, and these are submitted and approved by the Registrar of Companies.

The form of the Memorandum and Articles of Association is clearly set forth in any Companies Act, and your Sub-Committee followed out these instructions. An endeavour was made to retain and incorporate our present rules as far as possible, and they were taken in detail and an attempt made to build on additions. It was found that to carry on the project the majority of necessary rules would have to be entirely constructed on new foundations.

Explanatory clause 1 in the Memorandum of Association—"The term 'metals' denotes all metals and minerals other than coal."

Iron and coal in the older-established institutions are usually relegated to the province of the Civil Engineer.

As the iron industry is not yet fully established, it was deemed advisable to retain it under our denomination and to merely delete coal.

Some of our Members consider that coal should be included, but colliery administrators will doubtless determine the point when they are approached in due season.

Clause 2 does not call for comment.

Clause 3.—Objects—

(*a*) The present Institute carries on; it is merely reconstructed.

(*b*) A general clause not calling for comment.

(*c*) Incorporating our present functions in our new Constitution.

(*d*) A continuation of (*c*).

(*e*) New functions taken over by the Institute.

Throughout the new Constitution your Sub-Committee have endeavoured to link closer the educational and administrative branches of the profession.

Under the new Constitution, your Institute is intimately associated with every phase in the conduct of the profession.

(*f*) In its new capacity, as an administrative body, the Institute will have to enter the political field. An expression of opinion by the Institute will represent that of the whole profession in Australia and New Zealand.

(*g*) Another administrative function which does not call for comment.

(*h*) Ditto.

(*j*) Ditto.

(*k*) Extended powers are given the Institute in the establishment of sub-branch divisions and to provide for the establishment of sectional societies should the necessity arise.

(*l*) Does not call for comment.

(*m*) It appeared essential to your Sub-Committee that to establish the profession there must be some fixed prerogative relegated to qualified Members. The idea of a Charter did not meet with approval; qualification was the only attribute insisted upon, and provision was made to register all qualified men either within or without the Institute.

Some difficulty was experienced in choosing the branch of the profession governed by this prerogative, but it seemed essential that the issuing of a prospectus in connection with the flotation of mining properties claimed first attention, it being the point of our most intimate association with the general public.

It was felt that there should be several branches of the profession in which only the qualified man should hold that prerogative, but your Sub-Committee failed to define them; as suggestions come forward they may be incorporated.

It must be borne in mind that a prospectus must be signed by a qualified man. Anyone can make the report provided it is possible to obtain a qualified man's signature, and this signature will mean that, in the opinion of the Members of the profession in Australia, the prospectus in question sets forth an adequate representation of affairs as they appear to the profession generally.

No attempt was made to accurately define the obligations of qualified men in this respect, as it was felt that professional or unprofessional conduct would really be the ruling factor.

(*sub m*) In order to protect the interests of the profession generally this clause was inserted.

It does not greatly concern the large mining centres, but should be of considerable importance to the smaller ones. In this way the profession generally would always be represented as opposed to any individual section.

(*n*) Dealing with the Register.

The point to consider was whether the governing political bodies would delegate their powers to the Institute, and your Sub-Committee considered that, with a properly qualified representative body, they would readily confer the privilege.

Clause 4 deals with financial rules under our Constitution, and does not call for comment.

Clause 5. *Liability.*—In carrying out its Constitution, the Institute might have need of sums of money over and above the ordinary subscription. The limiting sum has also to be defined for the purposes of registration.

Taking our membership as 300, this would give us £7500 in this fund if it were needed. Our present membership is 473.

Clause 6.—Winding-up costs are covered by this clause.

Clause 7.—Necessary clause under our registration.

Clause 8. *Accounts and Their Audit.*—Another necessary clause under the terms of the Act, but not enlarged upon.

The subscribers to the Memorandum will be the Council at date of registration.

Now, dealing with the Articles of Association, or our internal rules—

1. No comment.
2. Our registration fees are defined numerically.
3. The only alteration is that of Associate Member to Junior Member.

In our public relations in Australia the term Associate Member is liable to be misconstrued, and your Sub-Committee deemed it advisable to alter the term to that of Junior Member.

4. Does not call for comment.
5. Merely defines the time and the nature of application under the proposed Constitution.

Qualifications.—The most important question under the proposed Constitution.

Your Sub-Committee considered that the most important factor in connection with the proposed Constitution was the certainty of recognition by such a body as the Institution of Mining and Metallurgy, provided that we came into line with regard to necessary qualifications.

Their established precedent in the main influenced the decisions arrived at with the one main exception; it was considered that sufficient time had

elapsed since the inception of our Schools of Mines and Universities to call for some recognition and inducement for technical training from our Institute.

To clearly define the attitude of the Institute towards the technically trained men, special provision was made in setting out his required qualifications, and it was thought that this departure would gain recognition from the I.M.M.

In the matter of service required, the applicant will have already graduated in the schools, and it was left to the discretion of the Council to determine the exact nature of the qualifying experience.

In the case of the non-graduate, the majority report of your Sub-Committee fixed the practical experience required at 7 (seven) years, to bring the time of service into line with that of the graduate who puts in 4 (four) years at the Schools and 4 (four) years' experience.

A minority report considered that a period of 5 (five) years' service in the capacities enumerated, included in a total mining service of 9 (nine) years, would be a sufficient qualification.

Your Sub-Committee had some difficulty in defining the Consulting or Reporting Mining Engineer, and fixed the number of properties reported on as a means of qualification.

It was considered that most Consulting Engineers would have already become qualified under other clauses.

Professor or lecturer, &c., for 3 years.

(*c*) *The writing of an original paper.*—It was thought that the writing of a paper could be taken as a standard of education, and credit should be given by remitting a proportion of the required practical experience.

(*d*) *Age.*—Following the established precedent of the I.M.M., the entrance age was raised to 30 years; the chief point considered was affiliation with the I.M.M.

It was also thought that the average case only should be considered, and that the average man would be 30 years of age before qualification.

At a subsequent report presented to 28 members of the Branch, the meeting was in favour of reducing the age to 25 years.

It was further suggested that to avoid contention the age limit could be eliminated from the proposed incorporation.

(*e*) Does not call for comment.

A Junior Member.—The graduate again is given encouragement, and the one year's experience was considered sufficient to ground him in the elements of practical application.

(*b*) In the case of the practical man, the four years is spent on the mine instead of at the Schools.

Your Sub-Committee endeavoured throughout to definitely fix all qualifications instead of leaving them to the discretion of the Council; the Council consists of professional men who cannot devote much time to Institute administration.

Student.—Will provide material for Members and Junior Members.

METHOD OF ADMINISTRATION UNDER THE NEW CONSTITUTION.

Your Sub-Committee incorporated in their first report some suggestions as to the composition of the Council.

Acting upon a suggestion from Mr. Palmer, they resolved to delete this portion of their report from the original, and tender it separately.

It was thought that the composition of the Council could be treated in the best manner by the present Council.

The principal points considered by your Sub-Committee as essential were that—

(1) The Secretary should be given extended powers, and should, in the main, be relied upon to manage the Institute and be afforded a ready means of reference to an Executive Council. He should be an official with a good salary.

(2) The educational bodies and the mining community should be brought into closer contact than at present, and representation should be given on the Council to the educational bodies.

That all liability due to membership on Council contracted by educational representatives, who are not otherwise Members of the Institute, should be taken over by the Institute.

That correspondence must take the place of meetings in connection with the major part of the Council's work, due to scattered members.

FINANCE.

There does not seem to be any reason why the Institute should not incorporate advertising matter in their journal.

It is hard to see why the medical associations think fit to advertise in their journal and make it self-supporting, whilst the mining associations do not.

This matter of making our journal self-supporting would entirely devolve upon the Secretary.

If we take our revenue as—

Subs.—300 Members at £3 3s.	£945
200 Juniors at £2 2s.	420
	£1365

we would have roughly £1365 per annum, all recoverable under our incorporation.

DIVISIONS AND BRANCHES.

In order to reach all Members in the outlying districts, it was proposed that the Council have the power of creating divisions in small mining centres.

Many Members in Western Australia would thus be reached who are at present too far distant from the Branch to participate in its administration. These divisions would send a representative to the Annual Meeting of the Branch when elections to Council were being held.

MEMORANDUM BY THE EXECUTIVE COMMITTEE.

The matter of incorporation and the wisdom thereof has frequently been discussed at meetings of the Council, but no definite scheme has, until now, been forthcoming. The scheme submitted by the Broken Hill Branch is a very elaborate one, and is evidently the result of considerable deliberation.

In the various Australasian States there is a growing tendency amongst technical institutes to incorporate. In Victoria, for instance, the Royal Victorian Institute of Architects, the Victorian Institute of Surveyors, and the Institution of Municipal Engineers of Victoria are incorporated. Amongst the incorporated institutes of Great Britain and America are:—The American Institution of Mining Engineers, the Institution of Mechanical Engineers, and the Institution of Civil Engineers (Lond.) The Institution of Mining and Metallurgy is not incorporated.

The benefits to be derived from incorporation are—

(1) The Association has a legal standing.

(2) The liability of Members is limited, and, except as members of the Association, Members of the Council are relieved of the total financial liability.

(3) The Association has the power to sue for moneys owing.

(4) No other Association can register under a similar name.

Until a Federal Companies Act is introduced, with the necessary provisions, it is doubtful whether incorporation should be attempted. The framing of such an Act has, it is understood, been mooted, but nothing further has been done.

Costs of incorporation vary. Where a draft of Memorandum and Articles of Association can be submitted to a solicitor for his revision and advice, the cost would vary from, say, £10 to £20. The registration fee under the Victorian Act is £5. and is possibly the same in the other States.

MEMORANDUM.

Name.—The name proposed is the "Australasian Institute of Mining Engineers." Although there are reasons for objection to departure from a name under which this Institute has grown, and under which it has become so well established, it has frequently been said that this name is not

in keeping with the qualifications of some of its most prominent Members. The tendency of this and similar Institutes is to embrace all branches of the mining profession, and, in a country such as Australasia, where the population is so small and where the number of members even of the largest Institutes must be proportionately small, compared to those of older countries, the wisdom of providing for all branches of the profession cannot be denied. A more comprehensive title should, therefore, be more appropriate. Titles which suggest themselves are—

"The Australasian Mining Institute"

"The Australasian Institute of Mines."

"The Australasian Institute of Mining and Metallurgy."

(1.) The proposal to exclude "coal" is not to be recommended. At present the Institute has amongst its members a number of eminent coal-mining engineers. The arguments put forward by Messrs. Power and Poole and Professor Waters are strongly in favour of the inclusion of "coal," and these arguments appear indisputable.

(3.) (*b*) This clause would be strengthened if, before the words "the production of metals," were inserted the words "the mineral industry and particularly." (This would then apply to some other clauses, such as (*d*).

(*d*) Professor Waters suggests the inclusion of the word "geology" after "mining and metallurgy." If it is not clearly understood that geology is a branch of mining this suggestion should be accepted.

(*e*) Is highly important. At nearly every First Ordinary Meeting of the Institute activities in the direction suggested have been advocated, both by leading members of the Institute and by other prominent mining men and legislators.

(*f*) Politics have always been "barred" by the Institute; but, in extending its scope as suggested in (*e*), an expression of opinion from such a body would be welcomed by legislators as a guide, and it is difficult to see why the Institute should remain silent whilst, perhaps, laws vitally affecting its Members and the profession generally were being discussed.

(*m*) This clause should form a link between the Institute and the general public, and should be welcomed by the mining investor as a guide against the promotion of "wild-cat" ventures. The clause should also include "coal" and "geologists."

(*n*) This clause would be indispensable if (*m*) is agreed to. The inclusion of "geologist" is necessary, as in (*m*).

(4.) This clause is imperative, otherwise registration would not be allowed by the Attorney-General.

(5.) Limits the liability of Members, in certain cases, to £25. This amount seems very high, and would probably prevent a number of very desirable men from joining the Institute, and consequently limit the membership (see Mr. Power's remarks).

ARTICLES OF ASSOCIATION.

Name.—This has already been commented upon under "Memorandum."

(1.) "Coal" should be included in this clause, as recommended in "Memorandum."

(2.) 1000 might be substituted in this clause for 500. The present membership of the Institute is considerably over 700, including "Members," "Associate Members," and "Students."

(3.) This clause defines the classes of Members, and recommends the substitution of "Junior Members" for "Associate Members." Both terms are somewhat misleading. The latter was chosen by the Institute after careful consideration. and is equivalent to the term "Associate" used by the Institution of Mining and Metallurgy.

(5.) This clause provides that each Member and Associate Member shall, before a date to be fixed, sign a claim to be entered under the new constitution.

(6.) *Qualifications.* — This clause defines very fully the qualifications necessary for each class of membership, and leaves little to the discretion of the Council. Mr. Poole, in his comments, suggests that these definitions should be elaborated in the By-laws. This suggestion has much to commend it, as it gives the Council some discretion. If elaborated in the By-laws and printed on the nomination forms the definitions (or qualifications) would be of great service to the Council and to Members wishing to nominate any person. The clause is the most important one in the Constitution, as the status of the Institute depends thereon.

(7.) *Election.*—This clause does not make it clear how elections are to be effected. Possibly the words "approved by them" were intended to mean "elected."

(9.) *Transfer.*—Although in (3) it is stated that "a Student shall not be deemed to be a Member," provision is made for his transference. The present practice of requiring a formal nomination as an Associate Member appears to be absolutely necessary.

(10.) Renews the issue of certificates.

(13.) *Subscriptions.*—The proposal to raise the amount of the annual subscription and entrance fee requires very careful consideration (especially under the present abnormal financial conditions).

Subscriptions to the Institution of Mining and Metallurgy are :—Members, £3 3s. ; Associates, £2 2s. ; Students, £1 1s.

(15.) Mr. Power's suggestion that a new member pay for the quarter in which he joins and for any subsequent quarter to complete the financial year has much in its favour.

(18.) Provision should be made in this clause (or in the By-laws) for certain notices *re* subscriptions to be forwarded to Members whose subscriptions are due or in arrears. These notices should be in the form of appendices.

(20.) *Branches.*—Elaborate suggestions are contained in these clauses for the establishment of Branches and of divisions of Branches.

(25.) This clause appears to be out of place, and should come under the next division, "Council."

Council.—This section recommends, amongst other things, that Past-Presidents should be members of Council. This, taking also into consideration other valuable recommendations in the same section, would mean a very large Council indeed. A limit might be fixed, to include Past-Presidents, say, for the preceding three or six years.

Provision is also made for the election of one member of Council by each Branch for every 30 members on the roll. This would mean that a State or Branch with less than 30 members would not be represented. The present method of proportional representation would act more equitably.

The representation of educational institutions on the Council, as recommended, used with discretion, should be to the interest of the Institute in many ways (see Mr. Power's remarks).

(36.) This clause deals with the powers of the Council, and, in the final draft, care would have to be taken that the provisions therein did not clash with section (3).

Financial.—This section recommends the securing of advertisements. It is necessary to point out that advertisements to the value of £80 a year are already attached to the Proceedings. These are from Victorian (principally Melbourne) firms alone. Whether other than mining advertisements should be included would have to be decided. Under any circumstances, even with the best magazines or journals, a trained canvasser is required to personally and persistently approach business firms. More frequent issues of the Proceedings would also probably be necessary if advertisements are to be made an important financial feature. It is very questionable whether the publications, printed up to the present standard, could be maintained by means of advertisements alone.

The estimated income, £1365, appears too sanguine, as allowance must be made for a considerable falling-off in the membership with the increase of subscription and the present financial position, due to the war. Although the Institute, under incorporation, would have power to sue for arrears of subscription, it would not always be advisable to do so, more especially as the mining industry is so subject to "ups" and "downs," and it is certainly not very flourishing at present.

Meetings.—No provision is made for holding General Meetings, and no suggestions are included regarding visits to mining centres—a matter of very great importance.

OPINIONS OF MEMBERS OF COUNCIL.

PROFESSOR D. B. WATERS :—

Memorandum of Association.—The name of the Institute should be The Australian Institute of Mines, and should include coal-mining as well as metal-mining, thus forming one comprehensive Institute for the whole mineral industry.

The terms "geology" and "geologist" should be associated with the terms "mining and metallurgy," "mining engineer" and "metallurgist."

Articles of Association.—*Qualifications for Membership (see Clause* 6*a*).—This clause is too wide; it should not be admitted that all degrees in engineering or science will permit of membership on an equal footing with degrees in mining or metallurgy. It would be necessary for the Institute to retain the right to say what degrees would be permissible.

Clause 6 (*b*).—Five years ample time, and under clause 2 (*b*) consider that 100 men for a metallurgical works is too many; a less number should suffice. In clause 3 (*b*) a qualification apart from reporting should be required. This clause seems unnecessary. Clause 4 (*b*) seems to imply that a professor requires to hold the position seven years before being entitled to full membership, whereas the holder of a professorship in mining, metallurgy, or geology in a University or mining school, recognized by the Institute, should *ipso facto* have full right to membership on application. Lecturers may be required to hold their positions for a period of years. Holders of the position of Inspector of Mines and members of Geological Departments should be provided for under this general clause of qualifications.

Clause 27.—Taking this in conjunction with clauses 24 and 32, it seems that an unwieldy body is aimed at, too large to expect a good attendance. In a State where there is no duly constituted branch the members would not be entitled to elect a representative. Clause 20 says "The Institute *may*" (not "*shall*") form branches. A system of State representation would be better—one or two from each State and New Zealand. Representation of educational institutions is, generally, to be commended, and Mines Departments should also be included; but if all these are asked to attend the Annual Meeting of the Council, the expense incurred would have to be borne by the Institute, and it is doubtful if the results obtained would justify such expense. Such Honorary Members could be consulted by local members of the Council, and, in regard to paying expenses, it would be better to pay the expenses of at least one member from each State to the Annual Meeting of Council.

Mr. Wm. Poole :—

(1.) The proposal to exclude "coal" is against the interest of the Institute and its members. Some classes of work receive keener attention in coal-mining than in metalliferous mining, and *vice versa*—*e.g.*, speaking broadly, and overlooking notable exceptions on both sides, ventilation, underground haulage, surface works and handling of output, methods for working bodies of material of low inclination, "breaking-down" of soft material, such as coal, shales, &c., distribution of power, are better undertaken in coal-mining, while methods of mining steeply-inclined bodies, removal of hard materials, drainage, &c., are better undertaken in metalliferous mining. The work of each section may beneficially influence the other—*e.g.*, the fine set of papers presented at the Wonthaggi meeting were of great interest to metalliferous mining engineers. The "Explanatory Notes," page liii, make use of the following:—"Iron and coal in the older institutions are usually relegated to the province of the civil engineer." The writers probably intend to convey an opinion quite different from the correct reading of this clause. Civil engineering, as defined in the Charter of the Institution of Civil Engineers (the mother of all engineering institutions), is "all engineering work other than military engineering"; hence the name.

(3.) No mention of Honorary Members.

(6.) *Qualifications.*—These should be defined in the By-laws or regulations under the Articles themselves, so that, under definite procedure, they may be more readily revised from time to time. This is approved general procedure. The qualifications would require careful consideration before being embodied even in the By-laws. The Council should retain a wide discretion as to what constitutes sufficient experience and training, both in quality and quantity. The proposal submitted from Broken Hill is too "cast-iron" and inflexible.

Election and Admission.—No mention is made as to how the election is to be made.

(7.), (8.), (9.) Should be in By-laws.

(13.) *Subscriptions.*—Should be in By-laws.

Council (26, etc.)—The basis for election of Council has much to recommend it. On the other hand, if strictly so carried out, it might leave the Institute without an Executive Committee, as, on the basis of one councillor for, say, 30 members, Victoria would have 3 representatives, perhaps only 2. The interests of the Institute would be better met by having a Council with a reduced number elected by the members as a whole (so as to ensure an effective Executive Committee), together with representatives from Branches and perhaps also from the principal teaching institutions. The Council should meet and transact business of policy during each annual session of meetings."

F. Danvers Power :—

Memorandum of Association.—*Clause* 1.—To win coal is mining. The distinction between coal and metal mining is artificial. There is much in common, and in future the working of large low-grade propositions will bring these two still closer together. There is nothing in coal-mining that a metal-miner could not soon pick up, and the differences are no greater than already exist between certain metal-mining methods.

Clause 5.—Why should an exception be made as to liabilities in connection with legal action ? It is hard lines if a member is made liable to £25 owing to an action of which he might not approve. Such a course is quite unusual in Institutes of this sort. The £1 mentioned in Clause 6 is sufficient, and if that was not enough to pay any claim then the Institute should go into liquidation and form a new body. Should any special cause arise requiring special expenditure for legal expenses in the cause of mining generally, then mining companies and others should be invited to contribute. Men will not join the Institute if expensive law suits are possible, and if not possible there is no necessity to make such a heavy provision for them.

Articles of Association.—*Clause* 6—" *Practical*."—It is supposed by " one or any " is meant one or a combination of any.

(36.) Here should be included " in the proportion of 30 reports in seven years," if any of this class of work is to count together with other classes. Even that means a little over four reports a year. Instances are known where mines took six months to be properly examined. Two such properties would take a man the full year, allowing nothing for travelling about. Surely such reports should be worth more than four smaller inspections.

Clause 15.—This does not read equitable. Anyone joining the Institute is presumably entitled to the Transactions for that year. but apparently anyone joining the Institute after 1st November gets the advantages of the balance of the year thrown in. If this view is incorrect, then the wording should be made so plain that there is no possibility of misunderstanding. The fair thing would be to make a new Member or Junior pay for the quarter in which he joins or for any subsequent quarter to complete the financial year. the minimum being the cost of a copy of the Transactions. As a Student's subscription about covers the cost of the Transactions, no allowance can be made in his case.

Clause 13.—Junior Members are here mentioned as voting in the matter of fees and subscriptions to be paid. Does not this clash with Clause 8, where Junior Members are not entitled to vote on any question affecting the Constitution of the Institute ? (*c*) Would it not be well to put an age limit to a candidate entering as a Student ? Otherwise a man of 40 years of age might join as a Student. If the idea is to let working miners join, so as to receive what benefit they can from the meetings and Transactions. then another word should be substituted for " Student."

Clause 27.—It seems that provision has been made for an unnecessarily large Council. Is it meant that Past-Presidents become Life Members of the Council? Why should Universities and Schools of Mines have representatives on the Council, who may not even be Members of the Institute? Such men should not have a vote in controlling the Institute's affairs, and if, as Honorary Members of Council, they have no vote, their position is obscure, for presumably the Institute is able to conduct its own affairs and know what it wants. Outside assistance can always be obtained if desired, and if the co-operation of the Universities and Schools of Mines is required for any purpose, it is not likely to be withheld if asked for.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY
FROM 30TH JUNE, 1915, TO 30TH SEPTEMBER, 1915.

Australian Mining Standard	weekly	Melbourne
Australian Mining and Engineering Review	monthly	Melbourne
Engineering and Mining Journal	weekly	New York
Iron and Coal Trades Review	weekly	London
Mining Journal	weekly	London
Mining Press	weekly	San Francisco
The Colliery Engineer	monthly	Scranton, Pa.
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry: Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa	monthly	Johannesburg
Franklin Institute: Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers: Journal	monthly	London
Chamber of Mines of Victoria: Monthly Mining Report		Melbourne
Chamber of Mines of Western Australia: Journal	monthly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines: Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines: Monthly Analysis of Production		Johannesburg
Rhodesia Chamber of Mines: Report of Executive	monthly	Bulawayo
Department of Mines, New South Wales: Mineral Resources, No. 19.		Sydney
Royal Society of New South Wales: Journal and Proceedings, Vol. XLVIII., Parts III. and IV.		Sydney
Department of Mines, Queensland: Geological Survey, Publications, Nos. 245, 247-249		Brisbane

Department of Mines, Western Australia :	
Annual Progress Report of the Geological Survey, 1914	
Geological Survey, Bulletin, No. 65 - -	Perth
Department of Mines, Tasmania :	
Geological Survey, Reports, Nos, 6 and 7	
,, ,, Bulletin, No. 20 - -	Hobart
Department of Mines, New Zealand :	
Polæontological, Bulletins, Nos. 2 and 3 -	Wellington
Institution of Mining and Metallurgy :	
Bulletins, Nos. 128-130 - - -	London
Institution of Mining Engineers :	
Transactions. Vol. XLIX., Parts 2 and 3 -	London
Iron and Steel Institute :	
Papers Nos. 1-13, 1915 - - -	London
Société Des Ingenieurs Civils De France :	
Bulletins, Nos. 1-3, 1915 - -	Paris
Société Géologique du Nord :	
Annales, XLII., 1913 - - -	Lille
Societa Geologica Italiana :	
Bulletin, Vol. XXXIV., Part 1 - -	Rome
Department of Mines, Canada :	
Geological Survey, Bulletin, Nos. 9, 15 and 17	
Annual Report, Mineral Production, 1913	
Summary Report, Mines Branch, 1913	
Preliminary Report on Bituminous Sands of Northern Alberta	
Lignite, Peat and Coal	
Report of Non-Metallic Minerals	
Petroleum and Natural Gas Resources of Canada	
Summary Report, Geological Survey, 1914 -	Ottawa
Mining and Geological Institute of India :	
List of Members, 1915 - - -	Calcutta
Smithsonian Institution :	
Annual Report, 1913 - - -	Washington
American Institute of Mining Engineers :	
Bulletin, No. 102 - - -	New York
Cuerpo de Ingenieros de Minas :	
Boletin, No. 81 - - -	Lima

RECENT ARTICLES ON MINING MATTERS.

(30th June, 1915, to 30th September, 1915.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

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(6) *The Engineering and Mining Journal* New York, wk, 15c.

(7) *The Colliery Engineer*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

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(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

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(28) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

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(30) *Proceedings*, Colorado Scientific Soc. Denver, Col.

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(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins*, Société des Ingénieurs Civils, Paris.

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(36) *Publications*, Iron and Steel Institute, London.

(37) *Proceedings*, Inst. of Mech. Eng., London.

(38) *Publications*, Field Columbian Museum Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

(41) *Publications*, Department of Mines, Wellington, N.Z.

(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(44) *Proceedings*, Geologists' Association, London.

LIST OF ARTICLES

ELECTRICAL.

Motor-Driven Duplex Pumps for Mine Service. E. P. Worden. (5) May 8, 1915.
Commutator Troubles Reduced by Slotting. Geo. E. Edwards. (5) July 17, 1915.
Electric Power for Montana Mines, Mills, and Smelters. Warren Aikens. (5) July 31, 1915.

GEOLOGICAL.

Correlation and Geological Structure of the Albert Oil Fields. D. B. Dowling. (29) Bull. No. 102.
The Geology of Prospects. F. P. Mennell. (35) May, 1915.
Annan River Tin-Field, Cooktown District, North Queensland: its Geology and Resources. E. Cecil Saint-Smith. (13) Series commenced Aug., 1914.
Geological Survey of the Cargo Goldfield. (11) M. Resources, No. 19.
Reconnaissance of the North Heemskirk Tin-Field. L. Lawry Waterhouse. (15) Geol. Surv. Report, No. 6.

MECHANICAL.

The Effect of Vacuum in Steam Turbines. G. Gerald Stoney. (37) No. 4, 1914.
Jigs and Fixtures. H. J. Thompson. (37) No. 4, 1914.
Pipe-Threading Machines for Mine Services. Geo. E. Edwards. (3) May 29, 1915.
The Distribution of Heat in the Cylinder of a Gas Engine. Prof. A. H. Gibson and W. J. Walker. (37) May, 1915.
High-Compression Oil Engines for Mine Service. Paul A. Bencel. (5) July 31, 1915.
The Prevention of Over-Winding and Over-Speeding in Shafts. G. G. T. Poole. (26) Vol. XLIX., Part 2.
The Winding-Drums of Practice and of Theory. H. W. G. Halbaum. (26) Vol. XLIX., Part 3.

METALLURGICAL.

Modern Gas-Power Blower Stations. Arthur West. (29) Bull. No. 102.
Conversion Scale for Centigrade and Fahrenheit Temperatures. Hugh P. Tiemann. (29) Bull. No. 102.
The Leaching Plant at Chuquicamata. Lincoln G. Rogers. (35) May, 1915.
Separation of Wolfram from Tin. M. T. Taylor. (35) May, 1915.
Surface Equipment of the Sons of Gwalia Mine, W.A., Describing Recent Additions Thereto. A. Wauchope. (42) June, 1915.
Analysis of Spelter. Wm. B. Price, Alden Merrill, Geo. L. Heath, Gilbert Rigg, Bruno Woeicie-chowski. (43) June, 1915.
Notes on the Practical Testing of Working Cyanide Solutions.
Wet Method of Mercury Extraction. E. Bryant Thornhill. (8) June 5, 1915.
Home-Made Apparatus for Cyanide Tests. Horace F. Lunt. (8) June 12, 1915.
Determination of Gold in Blister Copper. Rowland King. (8) June 12, 1915.
Preferential Flotation. O. C. Ralston. (8) June 26, 1915.
Degree of Crushing to Free the Economic Minerals. D. P. Hynes. (8) July 3, 1915.
Flotation at the Inspiration Mine, Arizona. (8) July 3, 1915.
The Statistical Position of Spelter and its Future. C. E. Siebenthal. (5) May 29, 1915.
Fumeless Ore Furnace Burning Crude Oil. (5) May 29, 1915.
Precipitating Action of Carbon in Cyanide Solutions. W. R. Feldtmann. (8) May 22, 1915.
Assay of Cyanide Solutions. C. E. Roodhouse. (8) May 15, 1915.
Concentrator of the Timber Butte Mining Co. Theodore Simons. (5) June 5, 1915.
Flotation in Australia. Charles A. Galbraith. (8) July 17, 1915.
Notes on Homestake Metallurgy. Allan J. Clark. (8) July 17, 1915.
Solution of Pulp Problems by Graphic Methods. W. J. M'Cauley. (6) July 17, 1915.
Electric Furnace at the Alaska Treadwell. W. P. Lass. (5) July 17, 1915.
Recent Progress in Flotation. O. C. Ralston and F. Cameron. (6) May 29, 1915.
A Modification of the Iodide Method. Arthur Fraser. (5) July 3, 1915.

Flotation in a Mexican Mill. (8) July 24, 1915.
The Position of the Tube Mill. Algernon Del Mar. (8) July 24, 1915.
Concentration of Gold in Bottoms in the Converter. H. F. Collins. (8) July 24, 1915.
Testing Working Cyanide Solutions. (8) July 24, 1915.
Arizona Copper Co.'s Dorr Thickener. David Cole. (6) July 24, 1915.
New Sampling Plant at Hamburg. (6) July 24, 1915.
A Rapid Method of Washing Gold Beads. E. J. Hall. (6) July 24, 1915.
Froth and Flotation. (8) July 31, 1915.
Notes on the Metallurgy of Zinc. E. H. Leslie. (8) July 31, 1915.
Notes on Sampling. T. W. D. Gregory. (26) Vol. XLIX., Part 3.

MISCELLANEOUS.

Some Remarks on Mining Education. Noah T. Williams. (26) Vol. XLIX., Part 2.
A Discussion of Hoisting Operations and Efficiencies. Letson Balliet. (5) May 22, 1915.
Stoping Methods at the Nevada Wonder Mine. Thos. M. Smitter. (8) May 15, 1915.
Technical Reminiscences. Albert R. Ledoux. (8) Series commenced May 15, 1915.
Development Methods at Fairbanks. Hubert I. Ellis. (6) June 12, 1915.
Discussing the Efficiency of Mine Lighting. Letson Balliet. (5) June 5, 1915.
Sorting and Breaking Ore in the Rand District. F. L. Bosqui. (5) June 5, 1915.
Arrangement for Ventilating Sinking Shafts, Winzes, and Drives. S. Nettleton. (5) May 8, 1915.
Balliet on the Benefits of an Adequate Mine Ventilating System. (5) May 8, 1915.
Indexing and Filing Technical Literature. Alvin R. Kenner. (5) May 15, 1915.
The Mother Lode Blast. F. S. Norcross, jun. (5) May 15, 1915.
Waste-Heat Boilers at Chrome. N. J. Clarence and L. Brower. (6) May 22, 1915.
The Prevention of Over-Winding and Over-Speeding in Shafts. G. G. T. Poole. (26) Vol. XLIX., Part 2.
Ore-Bin Construction. (8) July 24, 1915.
Copy of a Contract for Tin Ores Between European Smelters and Bolivian Miners. (8) July 31, 1915.
Friction Head in Water Pipe Lines. Geo. A. Ohren. (5) July 24, 1915.
Notes on the Occurrence of Petroleum in New Zealand. R. Speight. (23) July 12, 1915.
Preliminary Report on the Zinc-Lead Sulphide Deposits of the Rosebery District, Tas. Loftus Hills. (15) Geol. Surv. Report. No. 7.
An Auxiliary Aid Outfit for Attachment to Self-Contained Rescue Apparatus. M. M'Cormick (26) Vol. XLIX., Part 3.
Mining in Burma. C. W. Chater. (26) Vol. XLIX., Part 3.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 20.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

1915.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

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The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 Swanston Street, Melbourne, Victoria.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Abstract).

OCTOBER 4TH, 1915—1 P.M.

The Secretary's report was presented and accounts to the amount of £25 were passed for payment.

The application of Mr. Francis Grey Wilson for transfer from the class of Associate Member to that of Member was approved.

Mr. Isidore B. Fabrikant was admitted as a Student.

Resolved that special applications be made to members whose subscriptions were in arrears.

Other routine business was transacted.

NOVEMBER 1ST, 1915—1 P.M.

The Secretary's report was presented and accounts to the amount of £50, including Rent £12, and Publications £10, were passed for payment.

Nomination of Mr. E. Howard Greig as a Member was approved.

Resignation of Mr. V. G. Crawford was accepted.

Resolved that the Annual Meeting be held on Friday, 28th January, 1916, at 1 p.m.

Resolved that a list be prepared of Universities and Schools of Mines on the Faculties and Boards of which Institute representation was desired.

Other routine business was transacted.

DECEMBER 6TH, 1915—1 P.M.

The Secretary's report was presented and accounts to the amount of £47, including Rent £12, and Postage £12, were passed for payment.

Mr. B. H. Oxlade was appointed auditor of accounts for 1915.

The names of twenty members whose subscriptions were in arrears and who had received final notices, in accordance with the rules, were removed from the Register.

Authority was given for signing the draft copy of the Architects Registration Act, as amended.

Resolved that letters be forwarded to Universities and Schools of Mines requesting that the Institute be represented on Boards and Faculties controlling Mining Education.

Other routine business was transacted.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Application for admission to the Institute as a Member has been received from Mr. Ernest Howard Greig of Burma Mines Ltd., Nam Tu, Upper Burma. Mr. Francis Grey Wilson has applied for transfer from the class of Associate Member to that of Member.

ANNUAL MEETING, 1916.

Members are reminded of the Annual Meeting at Melbourne, on 28th January, 1916. The business of the meeting will include the presentation of the Annual Report and Balance Sheet and the election of officers.

INSTITUTE REPRESENTATION ON BOARDS AND FACULTIES.

Boards and Faculties controlling Mining Education in Australia and New Zealand have been requested to grant a direct representation thereon of the Australasian Institute of Mining Engineers.

SUPPLEMENTARY LIST OF MEMBERS WHO HAVE ENLISTED FOR SERVICE AT THE FRONT.

BENNETT, Lieut. V. C., Mining Corps.
COULTER, Lieut. LESLIE J., ,, ,,
DONALDSON, Major R. J., ,, ,,
HUNTER, Capt. STANLEY B., ,, ,,
MAWDSLEY, W. H., ,, ,,
NICHOLAS, FRANK H., Australian Expeditionary Forces.
ROSE, Lieut. WALTER J., Mining Corps.
SEALE, Lieut. H. V., ,, ,,
WALLMANN, Lieut. H. P., ,, ,,
WATERS, Capt. D. B., N.Z. Engineer Tunnelling Co.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY
From 30th September, 1915, to 31st December, 1915.

Australian Mining Standard	weekly	Melbourne
Australian Mining and Engineering Review	monthly	Melbourne
Engineering and Mining Journal	weekly	New York
Iron and Coal Trades Review	weekly	London
Mining Journal	weekly	London
Mining Press	weekly	San Francisco
The Colliery Engineer	monthly	Scranton, Pa.
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry: Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa: Journal	monthly	Johannesburg
Franklin Institute: Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers: Journal	monthly	London
Chamber of Mines of Victoria: Monthly Mining Report		Melbourne
Chamber of Mines of Western Australia: Journal	monthly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines: Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines: Monthly Analysis of Production		Johannesburg
Rhodesia Chamber of Mines: Report of Executive	monthly	Bulawayo
Royal Society of Victoria: Proceedings, Vol. XXVIII., Part 1		Melbourne
Royal Society of New South Wales: Journal and Proceedings, Vol. XLIX., Part 1		Sydney

Department of Mines, Queensland :
Geological Survey, Publication, No. 251 - Brisbane
Department of Mines, South Australia :
Review of Mining Operations to June 30, 1915 - Adelaide
Department of Mines, Tasmania :
Annual Report, 1914 - - - Hobart
Department of Mines, New Zealand :
Geological Survey, Bulletin, No. 17 - - Wellington
New Zealand Institute :
Transactions, Vol. XLVII. - - Wellington
Geologists' Association :
Proceedings, Vol. XXVI., Parts 3 and 4 - London
Institution of Mining and Metallurgy :
Bulletins, Nos. 131-133 - - London
Geological Society :
Quarterly Journal, Vol. LXXI., Part 1, No. 281 London
Geological Survey of Great Britain :
Summary of Progress, 1914 - - London
Institution of Mining Engineers :
Transactions. Vol. XLIX., Parts 4, 5 and 6 - London
North of England Institute of Mining and Mechanical
Engineers : Annual Report, 1915 - Newcastle-on-Tyne
Société Des Ingenieurs Civils De France :
Memoirs, Series 7, No. 4a, 6
Proceedings 1915, Nos. 4–7 - Paris
Societa Geologica Italiana :
Bulletin, Vol. XXXIV., Part 2 - - Rome
Sveriges Geologiska Undersökning :
Arsbok, 1914 - - - Stockholm
Geological Survey of India :
Record, Vol. XL., Parts 2 and 3 ; Vol. XLVI.,
Part 1 - - - - Calcutta
Mining and Geological Institute of India :
Transactions, Vol. X., Part 1 - - Calcutta
Bureau of Mines :
Bulletin, No. 25 - - - Toronto
Department of Mines, Canada :
Geological Survey, Memoir, 36, 67-69, 74, 78
Bulletin, No. 18
Products and By-Products of Coal - - Ottawa

Results of the Investigation of 6 Lignite Samples, 1915
Salt Deposits of Canada, 1915
Electro-Plating with Cobalt - - Ottawa

American Institute of Mining Engineers :
Transactions, Vols. XLVIII. and XLIX.
Bulletins, Nos. 103-107 - - - New York

American Society of Civil Engineers :
Transactions, Vol. LXXVIII. - - New York

California State Mining Bureau :
Mineral Production, 1914 - - Sacramento

Field Museum of Natural History :
Geological Series, Vol. V., No. 1 - - Chicago

University of California Library :
Rainfall of Berkeley - - - Berkeley

United States Geological Survey :
Water Supply Papers, Nos. 312, 331, 341, 343, 349, 350, 353, 365, 367 and 368
Bulletins, Nos. 559, 560, 563, 567, 582, 589, 594, 596 ; 340 F, 340 G, 340 H, 340 I, 340 J, 345 H, 345 I, 375 A, 580 L, 580 P, 581 E and 620 A
Professional Papers, 87, 88, 90 A, 90 J, 90 K, 90 L and 95 A
List of Publications, March, 1915 - - Washington

University of Illinois :
Bulletins, Vol. XII., Nos. 39, 40 and 47 - Urbana

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GEOLOGICAL.

The Geology of Kalgoorlie. C. O. G. Larcombe. (8) Aug. 14, 1915.
The Canbelego, Budgery, and Budgerygar Mines. E. C. Andrews. (11) Mineral Resources, No. 18.
The Catamaran and Strathblane Coalfields. W. H. Twelvetrees. (15) Geol. Surv. Bull. No. 20
The Wolfram Mines of Mount Carbine, North Queensland. Lionel C. Ball. (13) Pub. No. 251

MECHANICAL.

Operating Mining Plants in Parallel. W. Atkins. (5) Aug. 21, 1915.
The Superheating of Steam at Mine and Smelter Plants. C. A. Tupper. (5) Aug. 28, 1915.
The Hydraulic Compression of Air. A. E. Chodzko. (8) Aug. 14, 1915.
Prevention of Overwinding and Overspeeding in Shafts. G. G. T. Poole. (7) Aug., 1915.
Furnace Curves. R. J. Weitlaner. (3) July, 1915.
Blast Furnace Plant Auxiliaries and General Arrangement. J. E. Johnson. (3) July, 1915.
Boiler Corrosion. J. B. C. Kershaw. (32) Oct. 5, 1915.
Condensers for Evaporating Apparatus. (3) Sept. 1, 1915.
Installing and Operating Mine Power Plant Generators in Parallel. Warren Aikens. (5) Sept. 11, 1915.
Power Plant of the North Bulli Colliery, Coledale, N.S.W. R. C. Cliff. (32) Oct. 5, 1915.

MINING.

Stoping Methods and Drilling Problems on the Witwatersrand. E. M. Weston. (35) Aug., 1915.
The Stresses in the Mine Roof. R. Dawson Hall. (29) Sept., 1915.
A Description of an Improved Method of Removing the Broken Ore from Stopes. M. Weinbren. (28) Sept., 1915.

METALLURGICAL.

Roasting and Leaching Concentrator Slimes Tailings. L. Addicks. (29) No. 104.
The Commercial Production of Sound Homogeneous Steel Ingots and Blooms. E. Gathmann. (29) No. 104.
Lead Smelting at El Paso. H. F. Easter. (29) No. 104.
Zinc Dust Precipitation Tests. N. Herz. (29) No. 104.
The Hydro-Electric Treatment of Copper Ores. R. R. Goodrich. (29) No. 104.
The Advantage of High-Lime Slags in the Smelting of Lead Ores. S. E. Bretherton. (29) No. 104.
Slime Agitation and Solution Replacement Methods at the West End Mine, Tonopah, Nevada. Jay A. Carpenter. (29) No. 104.
Amalgamation Tests. W. J. Sharwood. (29) No. 104.
A Rule Governing Cupellation Losses. W. J. Sharwood. (29) No. 104.
Ore Handling System of the Arizona Copper Co.'s Smelter. C. A. Tupper. (5) Aug. 7, 1915.
Pertinent Points for Consulting Metallurgists. (5) Aug. 14, 1915.
High-Grade Slags in the Smelting of Lead Ores. S. E. Bretherton. (5) Aug. 14, 1915.
Metal Losses in Copper Slags. Frank E. Lathe. (6) Serial commenced Aug. 7, 1915.
Mechanical Ore Sampler. J. H. Taylor. (6) Aug. 7, 1915.
Mercury from Amalgamation Tables. E. B. Thornhill. (5) Aug. 14, 1915.
Leaching Copper Ore. L. Austin. (8) Aug. 7, 1915.
Cyanidation of Low-Grade Sulphide Ores in Colorado. H. C. Parmelee. (3) Serial commenced July, 1915.
What is Flotation ? T. A. Rickard. (8) Serial commenced Sept. 11, 1915.
Why is Flotation ? Charles T. Durell. (8) Sept. 18, 1915.
Ore-Dressing on the Mother Lode. E. S. Pettis. (8) Sept. 18, 1915.
Chemical Principles of the Blast Furnace. J. E. Johnson. (3) Serial commenced Sept. 1, 1915.

Callow Pneumatic Process of Flotation. (3) Sept. 1, 1915.
Magnetic Separation of Zinc Iron Sulphide Ores. (3) Sept. 1, 1915.
Efficiency of the Blast Furnace Operation. B. F. Burman. (3) Sept. 15, 1915.
Production of Zinc Oxide from Low-Grade Carbonate Ore at Leadville. (3) Sept. 15, 1915.
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Centimetres, cm. ; cubic centimetres, cc. ; metres, m.
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Square inches, &c., sq. in. ; cubic inches. &c., cub. in.
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Papers and Discussions.

CONTENTS.

MINING EDUCATION IN AUSTRALASIA.

BY D. B. WATERS.

THE Australian States and New Zealand have for many years provided instruction in mining, metallurgy, geology, and their allied subjects through their Universities and Schools of Mines. Though such education has been provided at considerable expenditure, there is no legal recognition whatever given to the holders of degrees, diplomas, or certificates granted by the authorities controlling these educational institutions. Certain of the Australian States and New Zealand require, however, that mine managers and other officials employed by the mining industry shall hold certificates granted by their respective Mines Departments.

The position at present is that each educational institution is practically a law unto itself, and therefore in many ways there is a want of uniformity amongst institutions all aiming at the same result—the training of mining men. The object of this paper is, in the first place, to point out certain aspects of this want of uniformity, and thereby how disabilities accrue. Some points in connection with the granting of State certificates will also be

[This paper is separately bound, and may be, if so desired, detached complete from this number.] A

noted, and then suggestions outlined whereby the present condition of things could be improved.

Mining Education Institutions.

It is impossible to mention all the Schools of Mines in operation throughout Australasia, but all such can be divided into three main groups, thus :—

(1) University Schools of Mines, or Faculties of Mines.

(2) Schools of Mines run on University lines, but not controlled by University authorities.

(3) Mining Field Schools of Mines.

The points to be brought out and compared are—

(*a*) The objects of instruction, as shown by the degrees, diplomas, or certificates granted.

(*b*) The regulations governing length of instruction, and practical experience required, for such above-mentioned qualifications.

UNIVERSITY SCHOOLS OF MINES.

In most of our Universities the term "School of Mines" is not employed—mining, metallurgy, or geology forming branches of the faculties of engineering or science; but this fact does not in any way affect the objects of instruction.

The following particulars of regulations bearing upon the points mentioned are taken from the most recently available calendars of our Universities :—

Melbourne University.

This University provides for the instruction of both mining and metallurgy, there being two courses open to students, viz. :—

(*a*) Degree course. | (*b*) Diploma course.

The degree course is only available to matriculated students, whilst the diploma course is available to either matriculated or non-matriculated students.

(*a*) *Degree Course.*—This is a course of instruction extending over a period of four years, and is a comprehensive course, including instruction in mining, metallurgy, geology, and the necessary allied subjects. The degrees granted are:—

(1) Bachelor of Mining Engineering (or B.M.E.).

(2) Master of Mining Engineering (or M.M.E.).

Besides attendance at classes and passing examinations, candidates require to do certain practical work before the degrees are granted.

For the B.M.E. degree candidates must put in two periods of five weeks each gaining experience with a civil engineer or surveyor, also a period of five weeks "in acquiring a practical knowledge of engineering in a mine or mining works." These three periods of five weeks each must be "put in" before the candidate enters upon his fourth or final year of instruction.

Further, before the degree is finally granted, candidates must produce satisfactory certificates showing that they have been engaged for twelve months in acquiring a knowledge of practical mining. In this connection it is stated that not more than three months of the twelve may be spent in mine surveying and draughtsmanship, not less than three months in underground mining under a competent mine manager or mining engineer, not more than four months may be spent in practical metallurgy under a competent metallurgist, not more than three months in geological surveying under a competent geological surveyor, and not more than three months in all in unskilled work.

The degree of Master of Mining Engineering (M.M.E.) is open to holders of the B.M.E. degree of one year's standing. This M.M.E. degree is granted under different conditions; thus, it may be granted by passing a special examination, or by the production of a thesis on a subject approved by the Faculty.

(*b*) *Diploma Courses.*—Two distinct diplomas are granted, viz.:—

(1) Diploma in mining. | (2) Diploma in metallurgy.

The course in each case is a three years' course, and candidates must also spend a certain time in acquiring practical experience.

For the diploma in mining the time necessary is twelve months, the conditions being the same as those governing the B.M.E. degree course. For the diploma in metallurgy, twelve months' experience is required, of which eight months shall be spent in purely metallurgical work under a competent metallurgist, and not more than four months at assaying under a competent assayer, or in research work, or the commercial treatment of ores under a competent instructor at a University, a School of Mines, or other approved institution.

Sydney University.

At this University there is no distinct Faculty of Mines; the subjects of mining and metallurgy are grouped along with mechanical, civil, and other branches of engineering under the Faculty of Engineering. The course provided is a degree course, and two degrees are granted:—

(1) Bachelor of Engineering (B.E.) in Mining.

(2) Bachelor of Engineering (B.E.) in Metallurgy.

The course in either case is a four years' one, and no practical experience is required except that two months' work is recommended.

Holders of the B.E. degree in either of these subjects may proceed to the Master of Engineering degree (M.E.) in mining or metallurgy, provided they are Bachelors of three years' standing, and have during that time pursued the practice of mining or metallurgical engineering. Also, all candidates for the Master of Engineering degree shall be required to pass an examination in some engineering subject, which he must submit for the approval of the Board of Engineering Studies, and also present a thesis embodying the result of an original investigation or design.

Adelaide University.

This University does not itself provide instruction in the technical subjects relating to mining and metallurgy, but has an arrangement with the South Australian School of Mines for such

instruction. Instruction in scientific subjects is given by the University itself. In this way duplication of work is avoided, and the arrangement is a good one, but there seems to have been a compromise in regard to the diplomas granted. Two courses of instruction are provided under this arrangement:—

(1) *Mining Course.*—This is a four years' course, and candidates must do twelve months' practical work in mining, of which time at least six months must have been spent continuously in one department of the industry before the diploma is granted.

(2) *Metallurgical Course.*—A four years' course, and twelve months' practical experience in metallurgical work is required under the same conditions as those mentioned for the mining course.

These two courses are only open to matriculated students of the Adelaide University, but the recognition is twofold.

The Adelaide University grants:—

(*a*) Diploma in Mining of the Adelaide University, officially styled Dip. M.E., for the mining course.

(*b*) Diploma in Metallurgy of the Adelaide University, officially styled Dip. Met., for the metallurgical course.

Whilst the South Australian School of Mines also grants to the same students:—

(*a*) Fellowship of the South Australian School of Mines in Mining.

(*b*) Fellowship of the South Australian School of Mines in Metallurgy.

These two courses are not to be confounded with the Associateship courses of the South Australian School of Mines, which will be considered later.

Brisbane University.

(Particulars not to hand).

Perth University.

(Particulars not to hand).

Tasmanian University.

This University does not provide for instruction in mining subjects, but recognizes certain work done by students of the Zeehan School of Mines, when such desire to enter the University and proceed to a degree.

New Zealand University.

The University of New Zealand is not itself a teaching institution, but is an examining, degree-granting body, and is a central institution to which are affiliated the four University Colleges of Auckland, Wellington, Canterbury, and Otago. It is only at the University of Otago that a Faculty of Mines exists, this Faculty being the Otago School of Mines.

The Otago University grants to students of its mining school diplomas of associateship in the following:—

(1) *Mining Course.*—This is a three years' course, requiring also twelve months' practical mining experience. Of this period not less than nine months must be spent in actual mining work, and, of the whole twelve months, at least three and not more than six months shall be spent in coal mining.

(2) *Metallurgical Course.*—A three years' course, requiring twelve months' practical experience in metallurgical and ore-dressing operations.

(3) *Geological Course.*—A three years' course, and students are required to spend six months in gaining experience in field geological work before the diploma is granted.

In any one of these divisions a student may obtain the Associateship of the Otago School of Mines, the official title being A.O.S.M.

In addition to these Associateship diplomas, there is also granted a "Certificate of Land and Mine Surveyor." This is a two years' course, and students are required to spend six months in the practice of land and mine surveying before the certificate is granted.

The New Zealand University also provides a distinct set of examinations in mining and metallurgical subjects, and grants

the degrees of Bachelor of Engineering in mining or in metallurgy, as the case may be. The scope of these examinations is based upon the curriculum of the Otago School of Mines, and the practical work experience regulations are the same.

Hence, in New Zealand there exists to some extent the same conditions as in South Australia, and a student can obtain for the same work two distinctions—namely, the A.O.S.M. of the Otago University and the B.E. of the New Zealand University. Each body in this case carries out its own examination, so that students taking both qualifications have to sit for two sets of papers on the same subjects. But only one body—the Otago University—gives instruction. The position in this case is anomalous, just as it is in South Australia.

MINING FIELD SCHOOLS RUN ON UNIVERSITY LINES.

The Schools of Mines which, from the information obtainable, come under this title are—

The Ballarat School of Mines.

The South Australian School of Mines.

The Charters Towers School of Mines.

There may be others, but information was not available.

Ballarat School of Mines and Industries.

This institution, so far as the mineral industry is concerned, grants associateships in—

(*a*) *Mining Engineering.*—A three years' course, with an optional fourth year. Candidates must also do twelve months' practical mining work under a mining engineer, mining surveyor, or mine manager.

(*b*) *Metallurgy.*—A three years' course, with an optional fourth year, and candidates must acquire twelve months' practical experience in metallurgical work, of which period at least three months must be served in the metallurgical laboratory attached to the school, and the remaining nine months at a metallurgical works approved by the board of examiners.

(*c*) *Geology.*—A three years' course, with an optional fourth year, and candidates must obtain twelve months' practical experience in some branch of work connected with the course under competent direction.

Besides these Associateship courses, certificate courses in assaying and mine managing are provided for, each being a three years' course, and candidates must produce evidence of practical experience to the satisfaction of the board of examiners.

South Australian School of Mines.

This school, apart from the granting of Fellowships in mining and in metallurgy in connection with its arrangement with the Adelaide University, and already alluded to, provides for courses of instruction for diplomas of associateship in mining and metallurgy.

(*a*) *Mining Course Associateship.* —A three years' course, and students require to gain twelve months' practical mining experience, of which at least six months must have been spent continuously in actual work in one department of the industry.

(*b*) *Metallurgical Course Associateship.*—A three years' course, the time of practical metallurgical experience being twelve months, and under the same conditions as for mining.

The official letters used for these associateships are A.S.A.S.M.

Charters Towers School of Mines.

This school provides for two associateship courses, viz. :—

(*a*) *Associateship in Mining.*

(*b*) *Associateship in Metallurgy.*

Each is a three years' course, and for each students are required to spend twelve months in gaining practical experience in mining or in metallurgy, as the case may be.

This school also grants a diploma as metallurgical chemist and assayer, the length of the course being three years.

MINING FIELD SCHOOLS.

This group comprises the majority of Schools of Mines, and such have as their objects the imparting of knowledge to those employed as miners and others in the industry, and also the instruction of those desirous of obtaining various certificates granted by the different Mines Departments. Some of these schools grant diplomas or certificates of their own, whilst others have no defined diploma or certificate course. Thus, in New Zealand there are mining field schools at Waihi, Karangahake, Thames, Coromandel, Westport, and Reefton, and none of them has the power to grant any "final qualification" for some definite course of study. This has ever proved a drawback to such schools, and one of the writer's ideas is to get over this disability. From what he has been able to gather, the same state of affairs exists in connection with other Schools of Mines throughout Australia.

On investigation it will be found that there are in some parts of Australasia too many Schools of Mines. This means that all are financially starved; the teachers are required to give instruction in too many subjects, whereby their efforts are dwarfed, and consequently students suffer. Each school should have as its object, as its privilege, and as its duty, the instruction of its students to some definite qualification, and its training should receive statutory recognition. Further, each mining field school should be so situated that some special branch of the industry associated with its own district can be the chief object of such school. This point will be again mentioned later on.

Position of Mining Education Institutions.

From consideration of what has been stated in regard to the "qualifications" granted by the various types of Schools of Mines, it is at once evident that the nomenclature assigned to the various degrees, diplomas, or certificates granted is as varied and intricate as the most whimsical of individuals could desire. Thus, in the

Australasian Institute of Mining Engineers there are men who are entitled to put after their names the magic letters B.M.E. or M.M.E., or B.E. or M.E., or Dip. M.E. or Dip. Met., or A.S.A.S.M. or A.O.S.M., and so on; and yet are not all such trained on the same lines, and from the same fundamental sources of knowledge? Why, then, this wonderful incongruity? Is it not possible that one reason why the members of the mining profession have never received statutory recognition of their training is this very incongruity of nomenclature, not only in Australasia, but everywhere else as well? The writer's suggestions aim at uniformity, such as exists in the professions of medicine, dentistry, law, and others.

Certificates Granted by Mines Departments.

The subject of mine managers' certificates was fully gone into at the Broken Hill meeting by Mr. Jas. Hebbard and the discussion thereon (Proceedings, No. 10, 1913), and certain points were brought out in connection with this subject—

(1) That the State should require all mine managers to be certificated. This the writer agrees with, since the States of Australia and New Zealand are spending money on mining education, and, further, it is only right that those who devote their time, energy, and means to acquiring knowledge should be protected and encouraged.

(2) That where a candidate has passed an adequate examination by some educational institution, such should be accepted by the State Department controlling the granting of the certificates. This is only right, as by doing so definite courses of instruction would be taken up by students, especially in mining field schools.

(3) The necessity for adequate and varied practical experience. It is on the subject of the amount of practical experience required that many of the mining education institutions and the State Mining Departments clash. In the writer's opinion, the educational institutions ask too little from their students in this respect, and nullify to their students the value of their diplomas by doing so.

The State is quite right in requiring adequate practical experience, though some amount of concession should be given to those attending definite courses of instruction at mining schools. The latest amendment to the New Zealand *Mining Act* (1908) permits holders of the Associateship in Mining of the Otago School of Mines to sit for a mine manager's certificate on completion of four years' practical work instead of five years required from others. It has, however, always been the writer's opinion that when a student of a School of Mines is granted a degree or diploma he should *ipso facto* become the holder of a mine manager's or other certificate. This will only be the case when the mining educational institutions recognize their duty, in respect to practical experience requirements, to their students, and if such practical experience were extended one would hear less about the cry of "stinking fish" from disappointed graduates, as the stern realities of the profession would be brought home to them and the misfits weeded out.

From a consideration of the various Mining Acts, it is shown that the various certificates granted by different Departments are :—

(1) Mine manager's certificates—first and second class—for both coal and metal mines.

(2) Certificates for battery and metallurgical works managers.

(3) Certificates for underground foremen.

(4) Certificates for mine electricians.

(5) Certificates for coal-mine deputies.

As regards the main ideas of examination requirements and practical experience required, there is considerable uniformity in the various regulations governing these different certificates, and yet they are not always interchangeable amongst the different States of the Commonwealth and New Zealand; whereas there should be no difficulty in bringing the various regulations into such agreement that a certificate granted by one Mines Department entitles the holder thereof to recognition by all other Mines Departments.

Suggested Improvements.

It is now self-evident that the want of absolute uniformity governing the regulations in connection with the "qualifications" granted by mining educational institutions, and of the "certificates" granted by Mines Departments, is due to the fact that the regulations are drawn up by a host of controlling bodies, all acting independently of one another.

The statement of the case is an easy matter, but it is certainly difficult to draw up a scheme of improvement, and the suggestions now made will doubtless not be acceptable to all; but, if any improvement is to be accomplished, it can only be by the action of one body acting for the whole mining community, and that body must be the Australasian Institute of Mining Engineers. The Institute has as its aim the gathering in of all men engaged in controlling the mineral industry of Australasia, and in order to make membership a necessity it must look after the material interests of its members by aspiring to such a position that its findings will be of weight in the community. The writer does not for one moment advocate that it become a political institution, or have as its main object the material betterment of its members, because then it would fail. Its main object must be always educational and the building-up of a literature on the very good lines already in existence, but still it must also assist members and prospective members to attain a status, and much of this can be accomplished by regulating their training and obtaining recognition for their qualifications.

The suggestions offered may be termed "The Standardization of Mining Education and Mining Qualifications."

The points to be specially noted are :—

(1) Qualifications for mining men. . (The term "mining" is used in its comprehensive sense, and covers all branches of work pertaining to the winning and treatment of metals and minerals. The term "qualifications" is used to denote what degrees, or diplomas, or certificates should be granted.)

(2) Courses of study required for different qualifications.
(3) Grading of Schools of Mines in regard to the work expected of each.
(4) Methods of examination.
(5) Practical work requirements.

"*Qualifications*" *for Mining Men.*

The suggested "qualifications" are :—

(*a*) *Diploma* or degree course of mining engineer, who shall be trained under the ægis of a University, and shall receive a diploma styled "Associate of School of Mines," the recognized official designation being A.S.M. This title is suggested because it is quite distinctive from any other title, especially M.E., and, further, it is probably the best known designation in the mining world.

The course for this should be a comprehensive one, embracing mining, metallurgical, and geological subjects ; but the candidates for this diploma, besides going through the general course, would be required to specialize either in mining, or metallurgy, or geology. In this way there would be produced men who, though all holders of the diploma of A.S.M., would yet be either mining engineers, or metallurgical engineers, or geologists. It is quite unnecessary to have a distinctive title for each of these—holders thereof would soon become known as to their special knowledge ; and, further, their several diplomas would plainly state which branch they were pre-eminent in.

(*b*) *Certificates*—

(1) Mining manager —(*a*) metal mining, (*b*) coal mining, (*c*) alluvial mining.
(2) Metallurgical works manager.
(3) Mining surveyor.
(4) Assayer and metallurgical chemist.
(5) Underground foreman.
(6) Mine electrician.
(7) Engine-driver.

In regard to Nos. 2 and 5, it would be necessary to divide these into groups. Thus, for instance, a metallurgical works manager may receive a certificate for ore-dressing, or copper smelting, or cyanidation; whilst an underground foreman may receive a certificate for metal mining, or coal mining, or alluvial mining.

Courses of Study.

The courses of study required for each of the foregoing qualifications should be uniform as regards minimum instruction and time in all schools of equal grade. This would mean that any school aspiring to a higher grade would have to furnish proof of its ability to provide instruction up to a given standard, and, further, its possession of the necessary laboratory equipment.

In the writer's opinion, the laying down of systematic courses of study is very important, because, especially in our University schools, will it be found that the old traditional academic ideas of Universities govern the advent of technical education into our Universities. In most cases mining students do not start their professional subjects early enough in their course, and such subjects are pushed aside by others of less importance. In our medical schools, for instance, the British Medical Association lays down the general ideas of the course, and that is what is required in the mining schools—somebody with authority to state what course of study shall be conformed to, so that such school may receive official recognition. The regulations governing courses of study would have to be flexible, otherwise the individuality of a school or of a teacher would be curbed, and that must not be. The object of some uniformity would help in attaining that recognition of schools of equal grade, of those from other schools, and also bring about that free trade in "qualifications" which is so desirable. Instruction itself is the most important part in technical training—much more so than the passing of examinations, and in this paper the need for systematic courses being laid down has been emphasized, but there is no intention to state here what these courses should be for the different "qualifications."

Grading Schools of Mines.

As already indicated, there exist at present Schools of Mines of different types, and so long as such institutions exist there will be differences between them in regard to the staffing and equipment, and also the calibre of the students available. To illustrate what is meant by this "grading," let us suppose that all existing schools are divided into three types, according to the grade of teaching each is capable of undertaking. These grades would be—

First-Grade Schools.—Comprising University schools and those in University type, all staffed and equipped to give instruction of the "diploma" course especially, and all or many of the "certificate" courses.

Second-Grade Schools.—Comprising those schools capable of giving instruction in the "certificate" courses, though to some extent these schools should specialize. For instance, a school in a coal-mining district would confine its instruction almost entirely to the training of coal-mine managers, coal-mine deputies or underground foremen, mine electricians, and engine-drivers. Again, a school in a gold-mining field, besides training metal-mine managers and underground foremen, would confine its energies to the training of gold metallurgical works managers, assayers, and so on.

Third-Grade Schools.—Comprising those schools capable of giving instruction for "certificates" as underground foremen, mine electricians, and engine-drivers, such schools being in most cases special schools.

The object of this grading is to give each school a definite object to work for, a definite course or courses of instruction, with the incentive to students that by going through a definite course they will attain some tangible "qualification."

Methods of Examination.

Mr. Jas. Hebbard, in his paper on "Mine Managers' Certificates," rightly pointed out that where a man had passed an examination—as, for instance, that set for a degree or diploma in

mining—it was not reasonable to ask him to again sit for examination by a State Department when such a one desires to obtain a State mine manager's certificate. One objection brought against such a course is that the diploma or degree examination papers are set by theoretical, non-practical men; whilst another objection put forward by State Mines Departments is that the examinations should not be passed until the full time of practical experience has been put in. Hence, we see that the diplomas and certificates granted by mining education institutions receive practically no recognition by State authorities.

With Mr. Hebbard's contention the writer is in thorough agreement; but still there is something for serious consideration by those controlling the educational institutions in the objections stated. The matter is capable of adjustment provided certain alterations are adopted on both sides in regard to the methods of examination and practical work requirements. It will be a very difficult matter to persuade State Mines Departments to forego their right of examination, but, in the writer's opinion, the difficulty can be overcome by co-operation in examination between Schools of Mines and Departments of Mines. Thus, in any one State let there be co-examiners for the diploma and certificate courses—one appointed by the State and one by the educational authority. In this way duplication of examinations would be avoided, and much advantage accrue to both sides.

Under present conditions, the obtaining of a State mine manager's certificate, for instance, is really based upon ability to pass a "technical" examination, provided the candidate has "put in" his practical time. The Department has no official knowledge that the candidate has received adequate instruction in the technical subjects, without which it is really impossible to gauge a man's knowledge by a short examination. Attendance at a definite course of instruction, whether it be for a diploma or certificate course, should be as essential a part of such diploma or certificate as practical experience requirements.

Each State could act independently so far as examinations are concerned; but, so long as the courses of instruction are uniform

for the different "qualifications," and also, of course, practical experience requirements, the value of the "qualifications" would be the same in all States, and mutual recognition would naturally follow.

Practical Work Requirements.

The amount of practical experience required before a "qualification" is granted is the real bone of contention between Schools of Mines and the various Mines Departments. Take the subject of "Mining," and it has been shown that in nearly all cases the Schools of Mines authorities require only twelve months' practical experience before granting candidates their degrees or diplomas, whereas State Mines Departments require five years' experience before candidates can obtain a first-class mine manager's certificate. In some instances a slight concession in time is granted to the holders of degrees or diplomas in mining. Still, the difference in practical experience requirements is very considerable. Which body is right is the question. For the writer's part, he supports, on the whole, the attitude taken up by the Mines Departments, and has to state that in his opinion it is absolutely wrong for Schools of Mines, more especially the University schools, to grant men "qualifications" of the highest grade where only one side of their necessary knowledge has been provided by such institutions. It is only right to students that when they receive the diploma of mining engineer that such should embody in it many of the certificates mentioned in the list of "qualifications," without further examination or further practical experience. The writer's meaning will be more clear from the following example under the proposed system. Suppose a student at a University School of Mines decides to obtain a diploma of mining engineer, specializing in metal-mining; he would then also have gone through the courses of instruction entitling him to the following certificates:—Mine manager (metal-mining), mining surveyor, underground foreman (metal-mining), mine electrician, and perhaps engine-driver. Also, the practical experience required of him should be of a length to cover the necessary experience

for these certificates. If, on the other hand, he decided to specialize in metallurgy, then he would receive also the certificates of metallurgical works manager and assayer. In order to effect this result, the length of practical experience required from those taking a diploma course in mining or in metallurgy should, in the writer's opinion, be four years. Supposing the course of instruction is a four years' one, then it might appear that a man would require to spend eight years in gaining his diploma. But such is not the case if the writer's scheme is followed out a bit further. Before stating it, however, it may be pointed out that in some of the Universities the degree courses (corresponding to the writer's suggested diploma courses) extend over four years, and after that students are required to "put in" twelve months' experience. This is not fair to the student; it puts him in an unfair position, since, having reached the status of a young man, with a University course behind him, he is forced to do lad's work, actually beneath his years, when he goes to work at a mine or metallurgical plant. His employer must necessarily, if going to pay him, make the best economic use of him, and can, therefore, very often only start him at lad's work, or on general labourer's work.

The scheme which should be adopted is one which will allow the student to gain his practical experience concurrently with that of technical knowledge. This can be effected by requiring attendance at classes at the University schools for a period of six months in each year, the remaining six months to be spent in gaining practical experience. Such practical experience should, if possible, be taken in districts where mining schools of the second grade are established, and which are schools for special instruction in some particular branch of mining or metallurgy, and the student should be required to attend a class or classes in such subject, such counting as part of his course.

In this way a student in a diploma course in mining or in metallurgy would, at the end of four years, have completed two years' practical experience, and would be in a position then to take subordinate responsible positions whilst completing the remaining two years required for his diploma as an A.S.M.

This whole time may seem long, but when you consider the case of a young fellow, say, of seventeen or eighteen years of age, it is not asking too much to require six years of him—in fact, even if the diploma were granted, say, at the end of four years, how many find that a wider experience is required before their diplomas are of use to them?

One important object of this system would be the fact that a man who obtained the diploma in mining or in metallurgy would *ipso facto* be granted the certificates pertaining to the particular branch he had obtained his diploma in, and the writer is quite sure that, unless the Schools of Mines authorities are prepared to require fully adequate practical experience of their students, no agreement with the State Mines Departments will be come to.

General.

It would make this paper too long to lay down a detailed scheme showing courses of instruction, practical work requirements, &c., for the different qualifications, and a scheme for grading mining schools and defining the special work of each in regard to coal and metal mining, metallurgy, and geology; but it has been the endeavour to carry this subject somewhat further than it has been by others, and point out lines upon which the Institute can act.

Schools of Mines are now recognized as an essential part of the mining industry, but it must be agreed that at present their position, on the whole, is unsatisfactory, and it is the duty of the Australasian Institute of Mining Engineers to take the matter up for the benefit of the industry as a whole and that of many of its members and prospective members. The subject is one of many difficulties, yet they are not insuperable. Mr. Jas. Hebbard, in the concluding remarks of his paper (already quoted), said:—"I will conclude with the hope that to-night's meeting will at least lay the foundation of a definite policy for placing the certification of mine managers on a sound and permanent footing." The writer's object has been to further Mr. Hebbard's hope, and give the start

to a definite scheme, though it includes more than the certification of mine managers. Considering the increase in metallurgical work which must come in these lands, our mining education system should receive the most thoughtful efforts for its furtherance by the Institute.

NORTH MINE PRACTICE IN UNDERGROUND POINTS AND CROSSINGS.

BY A. G. CAMPBELL.

IN a mine with any considerable output the economical handling and transport of ore underground is a matter of prime importance, and in the design or lay-out of a trucking system the question of what form the points and crossings are to take should be one of the first considered. Trucks may be moved from one line of rails to another by the following means: by flat sheets, with which may be coupled turn-tables, or by points and crossings. The latter class may be further divided into fixed and movable points. Flat sheets and turn-tables, owing to the time and skill needed by the trucker in using them, are seldom of great value with large tonnages, unless for purely temporary work, and the choice narrows down to one between fixed and movable points and crossings. Both systems have their disadvantages — movable points being liable to jamb with dirt and get out of order, whilst with fixed points trucks may sometimes run over the points and capsize, or go on to the wrong line, though this is not of frequent occurrence. On the other hand, the fixed points are simple, easily laid, and strong, and there is no necessity for the trucker to work any levers, the truck being guided on to its new direction by screwing.

Cast-iron points and crossings were first introduced into the North mine during the opening-up of the 950-ft. level, and the first arrangement was designed for a double turn-out in the main crosscut. Since then both the points and crossings have been modified several times, and it may be interesting to briefly outline the various forms through which they have passed, showing the reasons for the retention of some features and the rejection of others.

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The trucking lines have a gauge of 15 in., with 22-lb. rails, while the trucks hold, on an average, 1 ton of ore, and have a wheel base of 18 in. Fig. 1 shows the general arrangements of trucking lines in the main crosscuts from the 950-ft. level to the 1400-ft. level.

In the plan of the 950-ft. level (Fig. 1) the design is shown as originally put in; but the loop line at the end, which was placed there for handling steel to the magazine, has since been removed, so as

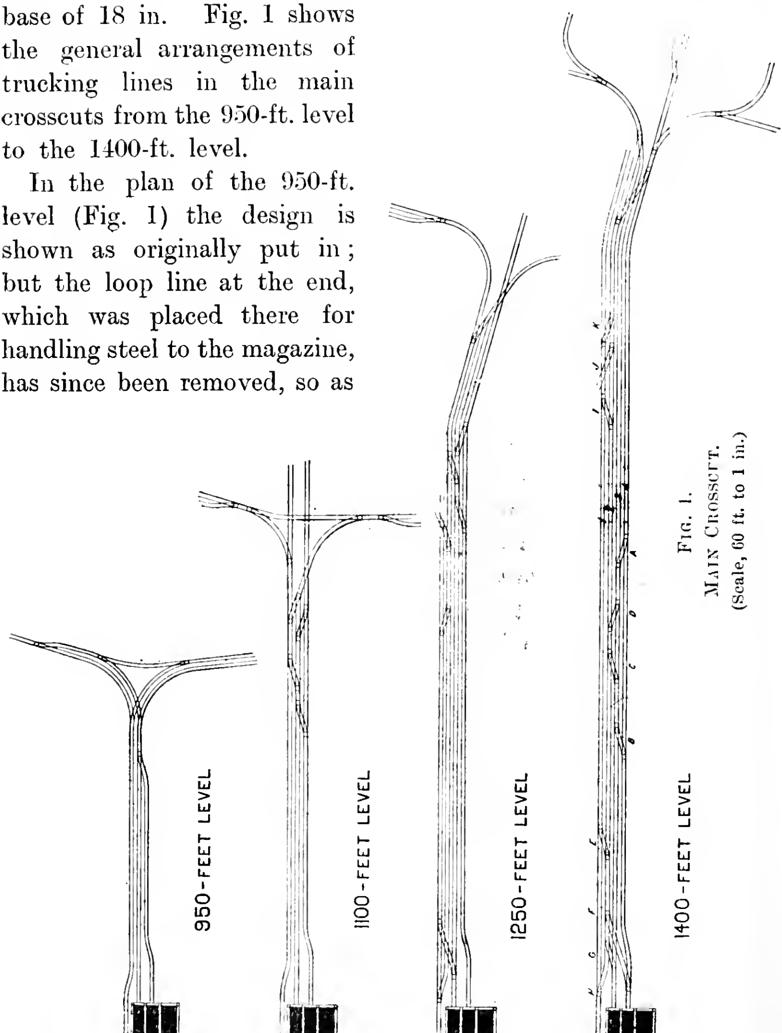

Fig. 1.
Main Crosscut.
(Scale, 60 ft. to 1 in.)

to cut out the open points at each end, the magazine being now behind the shaft.

These points formed a double turn-out for two lines, the radii being 20 ft. and 22 ft. 6 in. respectively, and were formed of shaped rails and guards with cast-iron filling pieces. They were made to no standard design, but simply to suit the particular conditions of the place, and, though they served their purpose well enough, they were discarded for the reason that any increase on these lines would have resulted in innumerable patterns and an enormous stock of spare parts to replace breakages. In addition to this, the excessive labour entailed in making them was another factor to be considered, as the following operations had to be gone through :—

(1) Laying out on a floor.

(2) Smiths—shaping rails to curves.

(3) Fitters—on rails.

(4) Pattern-makers—making patterns for frogs, &c.

(5) Fitters—fitting rails and filling-pieces.

For these reasons, this design was abandoned, and a plate somewhat similar to No. 1, but with replaceable points, was introduced. These points were blunt-nosed, of fitted cast iron, but were found to be easily broken and displaced by trucks; so forged mild steel was used in a practically similar design, but with no more success. One of the worst features of these was that they were liable to be removed and hidden or lost by truckers. Fixed points were then used, cast with the plate, and of equal length, but flat and obtuse; and these were afterwards changed—first to a plate with narrow points of equal length (No. 1 plate), and then to plates having long, narrow points of unequal length, and with less taper on the top. These last two plates are known as Nos. 2 and 3 plates, and, with No. 1, represent the standard design for points, No. 1 being used for trailing points only, while Nos. 2 and 3 are used as open points, the longer point being placed on the opposite side to the direction taken by the majority of the traffic.

In the same way, the frogs have been modified, until they appear as the frogs Nos. 1, 2, and 3, and the 90° and 60° crossings,

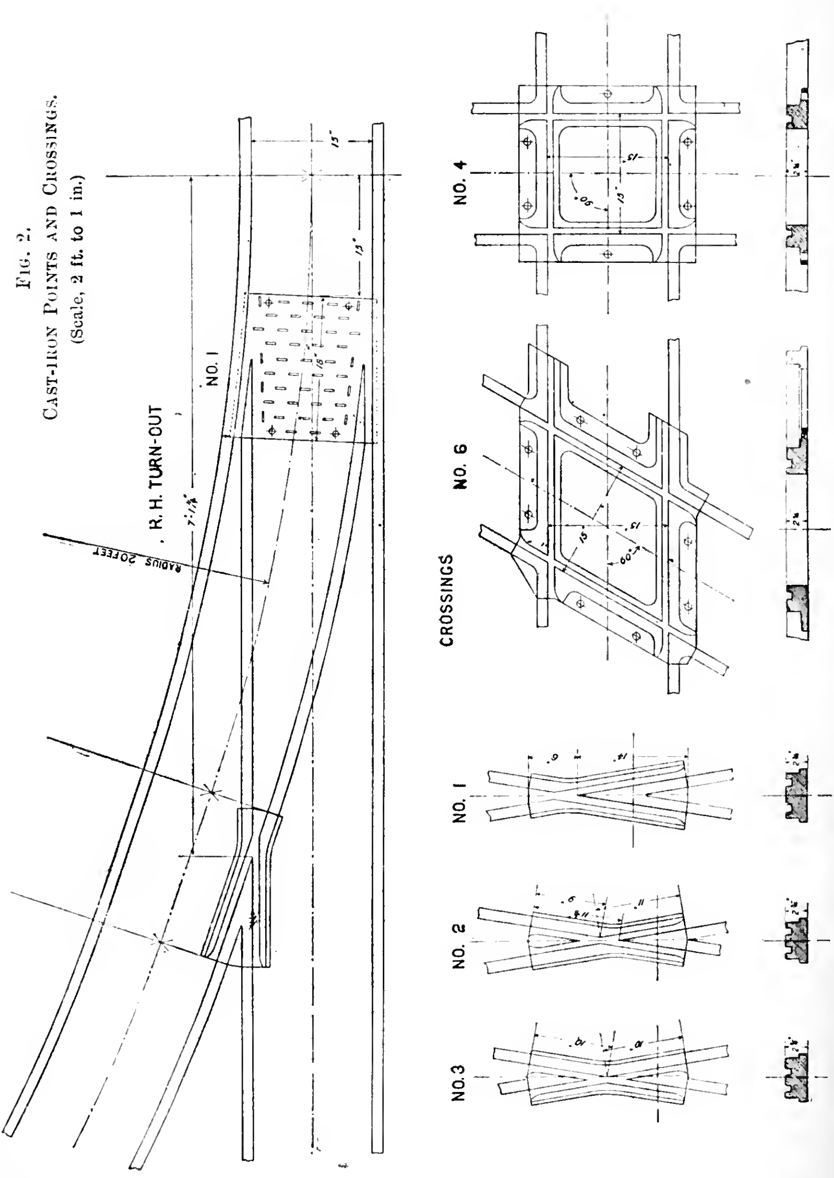

Fig. 2.
Cast-Iron Points and Crossings.
(Scale, 2 ft. to 1 in.)

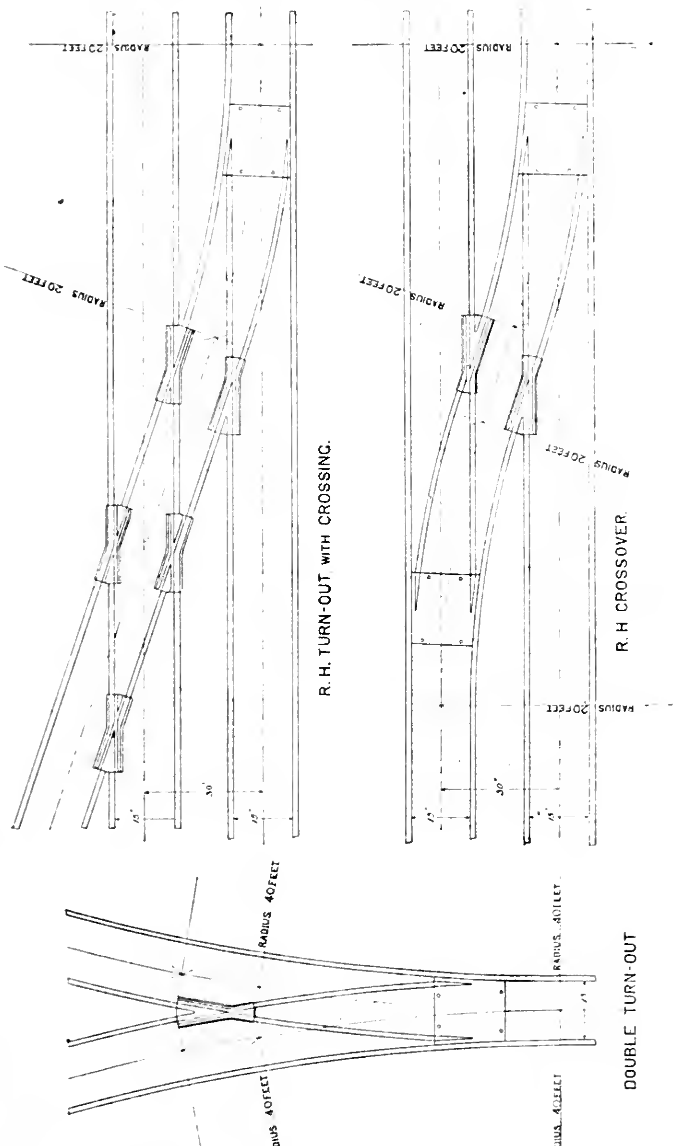

FIG. 3.
(Scale, $\frac{1}{4}$ in. to 1 ft.)

Nos. 4 and 6. The original intention was to cast them from Hadfield's manganese steel, but local cast iron was tried. and found quite satisfactory.

These designs show a great improvement on the originals on the 950-ft. level, owing to their being of standard design and radii, with frogs and plates always having a definite relation to the tangent points of the curves.

In the sketches are shown the general arrangements of four typical groups, viz. :—Right-hand turn-out, right-hand turn-out with crossing, right-hand crossover, and double turn-out. In each case the angle of crossing is 18° 40′. while the radii are all 20 ft., with the exception of the double turn-out, where the radii are of 40 ft., whilst the relationship of tangent points, plates, and points of frogs are shown. Both frogs and crossings are so strong and simple that renewals are infrequent. When renewals become necessary they are usually due to one of the following causes:—(1) Grooved points; (2) broken points; (3) worn points; (4) broken plates.

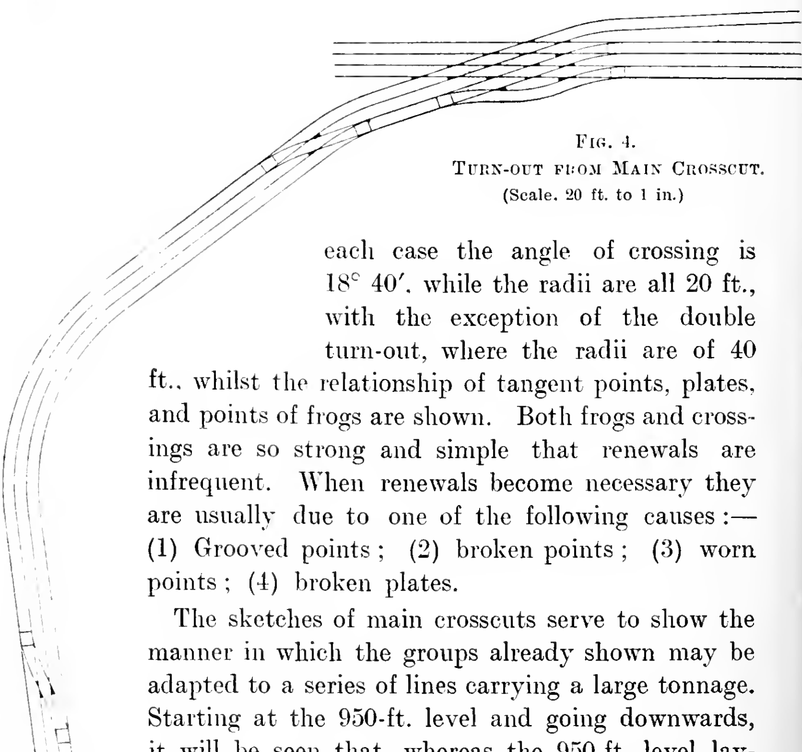

Fig. 4.
Turn-out from Main Crosscut.
(Scale. 20 ft. to 1 in.)

The sketches of main crosscuts serve to show the manner in which the groups already shown may be adapted to a series of lines carrying a large tonnage. Starting at the 950-ft. level and going downwards, it will be seen that, whereas the 950-ft. level layout is comparatively simple, on the other hand that of the 1400-ft. level includes a fair number of different groups. This

is due to the increasing tonnage and timber used in the larger width of lode in the lower levels.

In each case, except the 950-ft. level, the crosscut is shown going as far as the footwall drive, where the lines turn north and south and also go across the lode in a continuation of the main crosscut.

In describing the general design, it will be sufficient to take the 1400-ft. level, as this lay-out represents the latest ideas on the mine.

The functions of the lines may be divided into three classes :—(1) Handling ore ; (2) handling mullock ; (3) handling timber and stores. Ore and mullock are pulled on day and afternoon shift only, whilst timber is lowered during night shift. Ore and mullock come into the main crosscut from the north and south footwall drives or direct from the continuation of the main crosscut in the lode. In the case of the south footwall drives, it runs direct on to line 2, and in the case of the main crosscut and north footwall drive direct on to line No. 1. The full trucks then run down towards the shaft, and by means of the balancing crossovers at A and B the trucks may be run from one line to the other to balance the rakes. The crossovers at C and D were put in temporarily to provide for ore and timber during development, while permanent crossing, &c., were laid further on. These two will be ultimately removed at the crossovers at A and moved nearer to the shaft.

It will be noticed that advantage has been taken of the curve at the footwall drive to put in plates and frogs for curves into crosscuts, and past these curves the two lines will be connected by means of a turn-out in the same manner as in the 1100-ft. and 1250-ft. levels. Mullock from ordinary development work will be handled in the same way as ore, but, in the case of shaft mullock or development at the rear of the plat, as this is the lowest level at present, some provision has to be made. During sinking, the mullock is tipped into a bin at the back of the shaft, whence it is filled into trucks. No lines or shunts are used at the back of any plats, the empties being shoved out by the fulls

on to flat sheets, and then screwed and run out by the runner-out. By these flat sheets the trucks of mullock will be taken on to line 4, where they will be run along and temporarily kept till ready for pulling, when they will be taken on to the full lines, 1 and 2, by means of the crossovers at E, F, and G.

In handling empties, they are shoved out on to the flat sheets and then run on to either lines 3 or 4. Under ordinary circumstances they are run on to 4 and go along to the crossover K, where they run back on to line 3, and from there anywhere desired. If 4 happens to be blocked, they may be run on to 3 by the crossover at H.

Line 4 also serves as the stores line, and steel, &c., may be run on it to reach the magazine either by going direct on to 4 at G or E, or by going along 3 to I. No. 4 line ends in a dead-end past the magazine, which is used as a temporary depot for steel coffins, &c.

In the case of timber, the timber depot, when completed, will extend from the plat at the back of the shaft for a short distance at right angles to the main crosscut, and then parallel till it rejoins it by means of an oblique crosscut at J. All timber will go from the shaft to the back of the plat into the depot, where it will be run round and stored till needed, when it will be run out into the main crosscut by means of the turn-out and crossover at J.

From this the main idea of the lay-out of the other crosscuts may be gathered, it being only necessary to add that the loop lines on the 950-ft. and 1100-ft. levels have been removed owing to the trouble caused by open points previously referred to.

In laying the lines underground every care is taken that the foundation shall be as firm as possible. In laying the points for, say, a right-hand turn-out, the platelayers are given the distance of the point of the frog from some datum point. The straight rail is first dogged down temporarily and the frog dogged into position. The standard straight rail is then laid to gauge with the first rail, and the plate is then temporarily placed in position till the curved rail is dogged in its place, when the plate is fixed

by means of four 4-in. x $\frac{5}{8}$-in. coach screws passing through the holes shown.

This gives a general outline of the methods in use at the North mine in points and crossings. Like any other method, it has its disadvantages, but turn-outs, crossovers, &c., can be laid out so easily, and the points and crossings, when in position, are so little liable to damage or to wear out, that any inconvenience due to the necessity of screwing the trucks is more than compensated for. It may be mentioned that one of the plates on the 950-ft. level had 150,000 tons trucked over it before renewing, and, generally speaking, it is seldom that either plates or crossings need renewals.

NOTES ON LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME.

BY GEO. W. WRIGHT.

LAKE MARGARET is situate high up on the West Coast Range of Tasmania, and lies in a roughly triangular valley formed by the ridge of Mount Sedgwick on the south, the ridge formed by Mounts Geikie and Tyndal on the west, and a ridge connecting Mounts Geikie and Sedgwick on the east. Any description of this hydro-electric installation, however brief, naturally divides itself into the following sections :—

1.—STORAGE BASIN, &c.
2.—DAM.
3.—PIPE-LINES.
4.—POWER STATION.
5.—TRANSMISSION LINES.
6.—SUBSTATION.
7.—VARIOUS WORKS utilizing the power.

The *Storage Basin* used is the natural lake itself, which has a maximum length of 110 chains, a maximum width of 40 chains, and a mean area of 300 acres. The water-level of the lake, which has been maintained by a natural dam, is 2143 ft. above sea-level. The approximate catchment area at present appropriated is 7 sq. miles, and the rainfall, on the average, about 144 in. per annum. It is possible to increase this catchment area by suitable races, &c.

Dam.—The dam constructed is of very small dimensions, and is not a storage dam in the usual sense of the term. The natural storage of the lake itself was used by means of a channel cut through the natural dam and thence to the mouth of the lake, the lake level being thus temporarily lowered while the outlet pipes, screens, &c., were placed in position, and the channel was then closed again with a concrete wall in which the pipes and screens are set. This channel was cut partly through rock and partly through sand and conglomerate boulders for a total

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distance of 800 ft., giving the requisite depth of water. A large portion of the work entailed in this cut is now, of course, invisible.

The net depth of storage secured between spillway level and the top of the outlet pipes is 16 ft., and the total amount of storage required to carry the station over the driest period on record is only 7 ft., with a water consumption of 45 sec. ft. required for present loading.

The outlet pipes consist of two 48-in. internal diam. cast-iron pipes, set in the concrete wall. On the upstream side each pipe terminates in a screen compartment fitted with double screens of woven copper wire $\frac{3}{4}$-in. mesh, each screen being raised separately for cleaning. Between the lake and the screens flat sluice-gates are provided, to allow access to the screen chambers when same are closed. On the down-stream side the pipes are each provided with a 48-in. diam. sluice valve, operated through worm gear by hand, or by electric motor controlled from the power station switchboard. Only one of these outlet pipes is required at present to supply the station, the second one being provided for extension, but the pipes are interconnected, so that either can be used for the present service. It may be of interest to know that, excepting the sluice valves, the pipes, screens, sluice-gates, &c., were made in the company's workshops at Queenstown.

Pipe-Line.—The pipe-line from the lake to the power station consists of two distinct sections. The upper section, made of wood, is laid around the spur of a hill, on a flat grade, and really takes the place of the open channel common to so many hydro-electric installations, over which it has many advantages, obviating loss of water by soakage and evaporation, economy in water (as only such water as is required by the turbines from time to time is drawn from the storage), avoidance of *débris*, and allows of every foot of head being utilized. This section of the pipe-line is of wood-stave construction, having an internal diameter of 48 in., and the staves are 30 in number around the circumference. The wood used is seasoned Oregon pine, $1\frac{3}{4}$ in. thick, machined all over, with a V-grooved side joint and metal-tongue butt joint on the ends of the staves. The staves are held

together by mild-steel bands, $\frac{5}{8}$ in., $\frac{3}{4}$ in., and $\frac{7}{8}$ in. diam., according to the pressure. The total length is 7250 ft., total fall 45 ft., and the sharpest curve 150 ft. radius.

The pipe terminates in a rivetted steel header, at the junction with the steel pipe-line. This header is 6 ft. 6 in. internal diam. and 15 ft. 6 in. long, and has six openings, used for the following purposes :—

1 48-in. diam. inlet from wood-stave pipe.
1 48-in. diam. outlet to stand pipe for equalizing pressure.
1 29½-in. diam. scour pipe.
2 29¾-in. diam. outlets to steel pipes supplying station.
1 29¾-in. diam. outlet for steel-pipe extension.

The stand pipe is of similar construction to the main wood-stave pipe, and is 48 in. diam., and is carried up the adjoining spur 8 ft. above lake level. Local (King Billy) pine was used for this pipe. The wood-stave pipe was built under contract by the Australian Wood Pipe Company Ltd., Sydney.

The lower section of the pipe-line or pressure column is laid down a very steep mountain side between the steel header and the power station. The total length of the line on the slope is 2993 ft., and provided, together with the head in the wood-stave pipe and the lake storage, a total static head of 1100 ft.

The double steel pipe-line is in three sections of varying diameter—29¾ in. internal diam. at top, 26 in. in the centre, and 22 in. at the bottom section. The thickness of plates used varies from 7 mm. at the top to 14 mm. at the bottom, and are solid-welded throughout.

The muff or stuffing-box type of joint is used through the line, and solid-welded flanged joints on the distributing pipes, branches, &c., outside the station.

Each line is provided at the top, near the junction header, with a 29¾-in. sluice valve, hand-operated, and also electrically operated from the power station switchboard. A small air valve is placed at the head of each pipe.

The two steel pipe-lines outside the power station are joined together with a connecting casting, which is also used as a thrust

block for the end of the line, and, in conjunction with valves, allows either pipe-line to supply the turbines. Both pipe-lines are solidly embedded in large blocks of concrete at all changes of grade and direction, and each separate pipe is supported near the joint on a masonry pedestal. In order to keep a continuous record of water used, a Venturi meter is fixed in each of the distributing pipes.

The main-control sluice valves are 22 in. diam. and three in number, and are operated by a hydraulic cylinder, working off the pressure column, which, in turn, is controlled by pilot valves and a small electric motor put into operation from the power-station switchboard. These valves may also be operated by hand.

Power Station.—The power station is a concrete building, with steel roof trusses, galv. corrugated-iron roof, lined inside with fibro-cement sheets, and the main hall houses the four main turbo-generator sets and three exciter sets.

The main turbines are the Pelton wheel type, 500 r.p.m., with a normal rating of 1750 b.h.p., and a test efficiency of 84.6 %. The wheels are of solid steel, with polished steel buckets and cast-iron casing. Each machine is provided with two oil-ring bearings, tachometer, pressure gauge, and oil-pressure relay-type centrifugal governor, which is positively connected to a cut-off hood set over the water jet and through a dash pot to the control needle, which may be set to any desired closing time.

Each turbine is controlled by a hydraulic sluice valve, operated from the switchboard, and the speed of the turbines can be altered through a small electric motor attached to the centrifugal governor, also controlled from the switchboard. The main generators have a rated capacity of 1200 kw., at .8 power factor, and a voltage of 6600, and have a test efficiency of 95 %, and generate three-phase alternating current at a frequency of 50 cycles per sec. The small exciter sets are of similar design to the main sets, but run at 1000 r.p.m. Three of the main generating sets, fully loaded, are required to supply the works demand for power, the fourth machine standing by; also, two out of the three exciter sets are constantly in operation.

The various electrical divisions and units are controlled from a marble-panel switchboard, mounted on a gallery overlooking the main hall, having the following panels :—

3 exciter machine control panels.
1 Tirrill regulator panel.
4 main generator control panels.
2 outgoing transmission line panels.
1 local power and lighting panel.
2 valve control panels.

The switches are mechanically operated from the switchboard, but the switches are placed some distance away, so that no high-tension current is brought to the hand-operated switchboard. The whole of the switch gear, &c., is contained in a three-story annex to the main building. The lower floor houses the current and potential transformers, local power and light transformers. The middle floor accommodates the main oil switches, each set in a separate concrete cell, and the top floor provides room for the lightning arrester equipment, &c.

The pipes, valves, and turbines were supplied by Messrs. Boving and Co., London, and the electrical equipment by the Australian General Electric Co., Melbourne.

Transmission Line. — The length of the transmission line between the power station and the substation at Queenstown is 5 miles. Two separate lines, about 100 ft. apart, were erected for security of operation. The poles are round timber, spaced about 132 ft. apart. Each pole is fitted with two pine cross-arms with stays and crown-glazed high-tension porcelain insulators.

Each transmission line consists of three 19/12 stranded copper cables, and is provided with lightning arresters in duplicate at each end of line.

Substation.—The substation is a concrete and steel building, comprising a main hall and annex building for auxiliary gear.

The main switchboard is situated on the ground floor of the main building. This board controls the incoming main transmission lines, 6000 volts, the 6000 and 3000 volts sides of the main transformers, and distributes current at 3000 volts to the

North Lyell and Mount Lyell mines and blower plants, and also supplies current to transformers for the 550 volts distribution board. The switch gear is of similar design to that at the power station. The lightning arrester equipment is located in a gallery above the main switchboard.

The low-tension switchboard has been used for several years at the steam-generating station, and has been remodelled to act as a distributing switchboard controlling a number of the minor circuits throughout the works, as well as motor-generation sets and various subsidiary transformers. Another switchboard controls the low-tension D.C. circuits for battery and A.C. lighting circuits.

The main hall contains two 75 kw. motor generators for general lighting. A turbine is also being installed as an alternative drive for these generators, and will work off the smelter-supply water-main to provide light in case of a complete shut-down of the main scheme. Two other motor-generator sets are also installed to provide direct current for the electric crane in the proposed new converter plant.

The main transformers are of the single-phase, water-cooled type—nine in number—700 kw. capacity each, arranged in banks of three. Two banks are in constant use, and the third is kept as a reserve. A battery is installed for operating the relays on the main switches.

The high-tension electrical equipment of the substation was supplied by the Australian General Electric Co., Melbourne.

Smelter Blowing Plant.—Consequent on the power conversion from steam to hydraulic motors, it was necessary to instal an electrically-driven blowing plant for furnaces and converters.

Two high-tension circuits run from the substation to the blower house, which is a steel and concrete structure, and, operating through suitable control panels on the blower switchboard, drive six motor-driven centrifugal blowing engines. Four of these engines are used to supply air to the smelter furnaces—three on full load and one spare machine—and the other two engines supply air to the converter plant. The smelter blowers have a capacity

of 25,000 cub. ft. free air per min., at pressures varying from 4 to 6 lb. per sq. in., and the driving motors, 850 h.p., are ventilated by separate motor-fan sets, one to each blower. The converter blowers have a capacity of 3500 cub. ft. free air per min. to a pressure of 12 lb. per sq. in., and the 250 h.p. motors are self-ventilated.

All the blowing engines run at a speed of 3000 r.p.m., and are fitted with tachometers, pressure gauges, &c., and the bearings are oil-ring lubricated and water-cooled.

The whole of the blowers and switch gear complete were manufactured by Messrs. Brown, Boveri and Co. Ltd., Baden, Switzerland.

Other uses to which the power derived from the hydro-electric scheme is applied are—

The Mount Lyell mine—hauling, winding, air compression, and lighting.

The North Lyell mine—hauling, winding, air compression, and lighting.

Converter Department—general power, lighting, &c.

Valley Pumps—for water supply.

Workshops—generally.

Slag Pumps at Smelters.

Town and Works—lighting generally.

Flue Dust Plant.

Main Haulage—for ore transport to reduction works.

Howard's Plains—firewood haulage.

The extension of the railway line to the power station was started in July, 1912, and the work in connection with the hydro-electric scheme in February, 1913. The water was turned on to the turbines on 8th November, 1914.

[An amplification of these notes will be published in a subsequent issue of Proceedings.]

THE APPLICATION OF SURFACE COMBUSTION.

BY J. H. LEDEBOER.

FROM time to time articles have appeared in the technical press on the interesting subject of surface combustion. The theoretical part of the problem has, for practical purposes, received ample consideration in those articles, but the number of practical applications of surface combustion referred to has generally been very small.

It is the writer's object to review some recent developments in surface combustion, and also to try to arrive at an estimate of its advantages and possibilities.

It is not necessary for this purpose to dilate upon the phenomenon of surface combustion itself. Many interesting things have been said on the subject by Prof. W. A. Bone, of Leeds, to whose researches the recent developments are largely due. A few illustrations may make clear what is meant by surface combustion.

Prof. Bone, in an experiment, took a platinum crucible, which he heated in the flame of a Mèker-burner. Then he turned the gas off for a second, and, opening the stop-cock again, allowed the combustible mixture to impinge upon the crucible, the temperature of which was still well below the ignition temperature of the gas. Although no flame was visible at this stage, it was soon clear that, nevertheless, the gas and air had continued to combine, as the crucible would attain a red heat after a few seconds.

Another instance of surface combustion was shown by a Mr. Fletcher, in 1882. He placed an iron ball, weighing 3 lb., on a slab of fireclay, heated it for a few seconds with a blow-pipe, and blew the flame out, leaving the gas-mixture still impinging on the

[This paper is separately bound, and may be, if so desired, detached complete from this number.]

ball. The room was darkened, but no flame was visible. After a few seconds the ball became so hot that it began to melt.*

In both these instances surface combustion takes place at a temperature well below the ignition temperature of the gas mixture. Apparently, the cause of this is in the accelerating influence which hot surfaces exert upon chemical changes in gaseous systems.

That surface combustion and ordinary flame combustion are two different phenomena may be gathered from the following facts :—

(1) Surface combustion can take place at a temperature well below that at which flame combustion becomes possible.
(2) The rate of combustion is very much higher than with flame combustion, and consequently much higher temperatures can be attained.
(3) There is no visible flame.
(4) The presence of water vapour accelerates flame combustion, while it retards surface combustion.
(5) With flame combustion methane has a greater affinity for oxygen than hydrogen has ; with surface combustion the reverse is the case.

The simplest application of surface combustion is diaphragm heating. For this purpose Prof. Bone made a diaphragm of fireclay, crushed to pass a $\frac{1}{8}$-in. mesh and to be retained by a mesh of 1/16 in. This material is mixed with a fine meal of felspar, moistened with water, dried and baked in a kiln.† The result is a slab of a very porous texture. It is quite easy to blow through one 1 in. thick.

A slab of this description is fixed in a cast-iron frame (Fig. 1), forming a flat chamber with a connection at the back to supply an explosive mixture of gas and air. The supply of air can be adjusted to almost the theoretical quantity.

In order to start the reaction, gas only is admitted at first. This burns with a flame on the surface of the diaphragm. After this

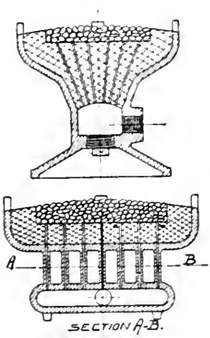
* *Engineering*, 1912, p. 657.

† *Engineering*, 1911, p. 488.

air is turned on, the result being that the flame is blown out; but surface combustion sets in, and soon the surface of the slab is at red glow, all the gas being burnt at a rate much higher than is possible with flame combustion. The glow is apparently limited to a surface layer about $\frac{1}{8}$-in. deep.

The temperature of the face of the slab, other things being equal, depends on the pressure of the explosive mixture, and

FIG. 2.



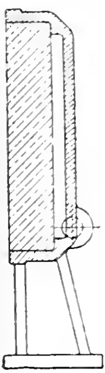


FIG. 1.

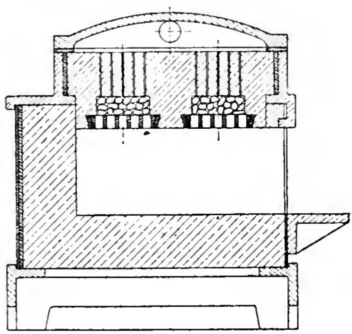


FIG. 3.

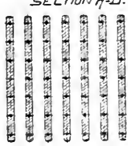


FIG. 4.

responds so readily to any variation in this pressure that there is no perceptible lag in the temperature.

It should be noted that outside air does not take any part in the process of combustion. A diaphragm can be lowered in CO_2 or N_2 without being affected in any way.

These diaphragms can be used for various purposes, their main advantages being in the high temperatures attained, and easy application of the heat in any direction to a limited surface. They are, for instance, very convenient for further concentration of highly concentrated solutions, as the heat can be applied to the surface of the liquid by arranging the diaphragm face down.

As long as the heat is allowed to radiate freely, the back of the slab casing will be quite cool; the velocity of back-firing at this temperature is low, and, the velocity of the explosive mixture in the pores of the slab being higher, back-firing does not set in. If, however, the radiation of the heat is obstructed, the temperature of the diaphragm will rise, and the velocity of back-firing, rising with it, may, at the place where the gas enters the slab, attain a point at which it is higher than the velocity of the gas, and back-firing will take place. This would prevent utilizing one of the typical advantages of surface combustion—*i.e.*, the high temperatures attainable with very simple means, these temperatures being about 2700° F. with Mond gas, and probably over 3600° F. with coal gas. It is quite possible that temperatures even higher than these could be attained, but the difficulty is in obtaining refractory material that will stand it.

Now, with a porous diaphragm sufficiently high velocities of the gas to prevent back-firing can hardly be attained at such high temperatures. The available area for the passage of the gas is too big; and, if this area was sufficiently restricted to obtain the necessary velocity, the frictional losses would become excessive.

These difficulties are overcome by using, instead of a porous diaphragm, a loose mass of granular refractory material—for instance, broken firebrick—the fragments being of about hazelnut size. The explosive mixture can be supplied to this bed of refractory fragments by means of small-size holes, which are specially arranged for the purpose.

Fig. 2 shows a burner made in this way. The holes for the passage of the gas are long enough to keep the lower end cool. These holes are about 1/16 in. diam., and back-firing does not occur even after turning the gas mixture off. We have as an available acting surface the total area of the fragments.

In cases where the heat has to be applied in a horizontal or downward direction, the fragments can be prevented from dropping out by inserting a grate of refractory material. Fig. 3 shows an application of this to a small furnace.

To reduce the size of the burners, air-gaps can be provided for

cooling. The canals can then be made much shorter. Fig. 4 shows a burner made in this way.

A bed of refractory fragments can also be used with advantage for the heating of all kinds of furnaces and muffles.

Fig. 5 shows a section of a furnace in which lumps of fireclay are arranged all around the floor. Several nozzles are used to admit the gases, the size of hole and length of nozzle being determined with a view of preventing back-firing. These nozzles can be quickly replaced, if necessary. It is claimed that with this

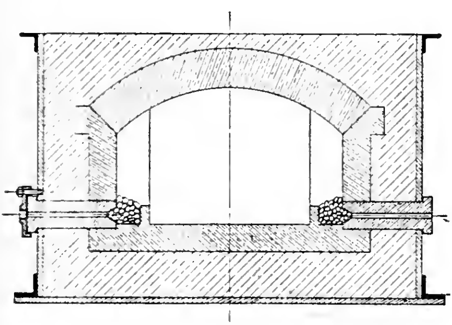


Fig. 5.

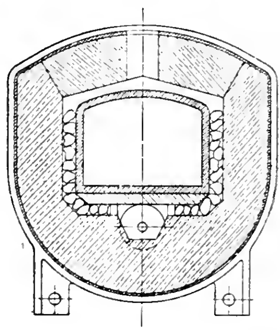


Fig. 6.

arrangement of furnace the temperatures are remarkably uniform, the variation on an average temperature of about 1800° F. being not more than 45°–55° in different places.

Of late, designs have been simplified by reducing the number of nozzles. Fig. 6 shows a muffle furnace arranged with a single nozzle. This furnace is apparently constructed for the use of liquid fuel, to which reference will be made later on. The space between muffle and outside wall has to be only slightly larger than the size of the fragments, so that these are practically arranged in a single layer.

Surface combustion has also been applied to steam boilers, fire-tube boilers being best suited for this purpose. The tubes are fitted with an inlet nozzle, as shown in Fig. 7, through which the explosive mixture is introduced. This nozzle not only prevents back-firing, but also protects the boiler front from excessive

temperatures. The rest of the tube is filled with refractory fragments.

It is important to note that the combustion is completed in the first four or six inches of the tube, and that the remainder of the fragments serves an altogether different purpose, which has nothing whatever to do with surface combustion. This was also the case in the muffle furnace shown in Fig. 6. The idea is that the fragments act as baffles and cause the gases to mix continually and to impinge repeatedly on the tube-wall. In this

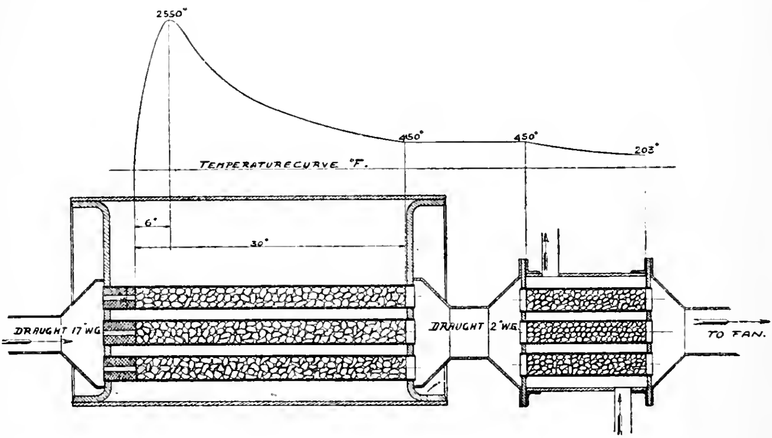


FIG. 7.

way no cold insulating layer can be formed near the tube-wall, preventing the much hotter core of the gases from giving off its heat.

The result of this baffling and mixing is startling. With a 3-in. test tube 3 ft. long, the gases, which reach a temperature of about 2550° F. within 6 in. of the inlet nozzle, would be cooled down to 340° at the outlet end. The average evaporation per sq. ft. of heating surface is enormous, being from 22 lb. to 30 lb. per hour. This is five times as much as for an ordinary boiler, and three times as much as for a locomotive boiler.

The gases, after leaving the boiler, can be further cooled down in a feed-water heater, the tubes of which are also packed with

fragments. Although the heater shown in Fig. 7 is very small, the gases are cooled down to 203° F. with a feed-water temperature of 136° and a cold-water temperature of 42°. Even this can be improved upon.

The tubes of the boiler shown in Fig. 7 are not even 3 ft. long. From the point of view of thermal efficiency, there is no reason why a boiler should be made longer, no matter how large its output, because a 3-ft. tube does all that is required. The whole arrangement is exceedingly compact and simple, and a boiler of this description should be cheap. High temperatures are kept well away from the ends, so that temperature stresses are small. The whole of the outside of the boiler can be very conveniently insulated, and radiation losses thereby reduced to a minimum.

It is claimed that in a boiler of this description 94 % of the available heat is given off to the water.

A drawback of the system is the considerable suction necessary to draw the gases through the filled tubes. With the boiler shown in Fig. 7 this suction is about 17 in. W.G. (water guage), of which 15 in. is used in the boiler and 2 in. in the feed-water heater. It would seem that the power used for this purpose would be considerable, but this is not so. It takes about $2\frac{1}{2}$ % to $3\frac{1}{2}$ % of the total output of the boiler, so that the over-all efficiency would be well over 90 %. The total quantity of the gases is reduced to a minimum, there being hardly any air-excess, and this accounts for this low power consumption.

It is necessary, with such high draught, to have air-tight flues. Brickwork would not be suitable for this purpose. Also, in this respect, the fire-tube boiler is best suited.

In the first experimental boilers the gas mixture was admitted to a chamber arranged in front of the boiler under pressure, resulting in a greatly increased velocity of back-firing at the given temperature. Screens, similar to those of a Davis lamp, but made of finely-crushed refractory material, had to be arranged to prevent this occurrence. In order to be able to do away with these complications an arrangement was adopted by which the gases are sucked through the boiler. This means a considerably

lower average pressure of the gases, and therefore bigger volume and increased frictional resistance, also a lowering of the temperature of combustion, which is an advantage, because it tends to reduce the very violent boiling-up at the hottest part of the tubes. Another advantage is that now observation holes can be arranged in the front of the admission chamber, there being no screen to block the view, and each tube can be regulated, the glow of the fragments being a sufficient guide.

A further simplification was obtained by the adoption of a small number of tubes of large diameter for the combustion of the gas, while a greater number of small tubes, through which the products of combustion are subsequently passed, make up the necessary heating-surface. The inlet nozzles for these large tubes have several holes for the admission of the gas mixture. The whole arrangement in front of the boiler is thus made much simpler, and the supervision easier. At the same time the too violent formation of steam at the hottest part of the tubes is again reduced by reducing the extent of this hottest heating-

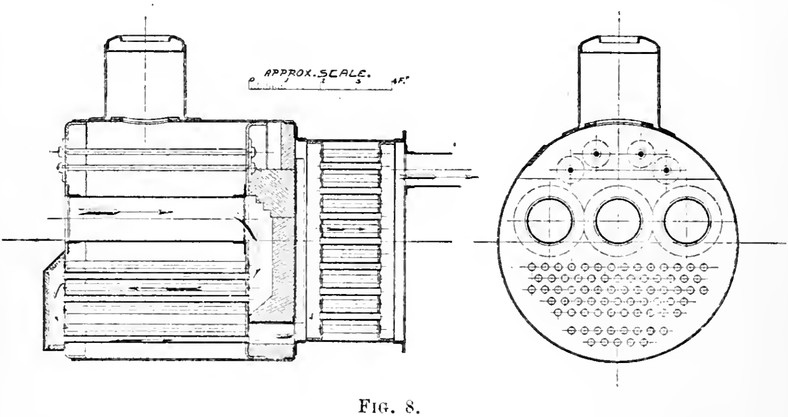


Fig. 8.

surface. The big tubes are arranged on a higher level than the small ones, so that the majority of the steam will have to pass a minimum thickness of water. This tends to make the steam less wet, and to provide a comparatively quiet corner near the bottom

of the boiler where sediments can separate out. The formation of steam on the outside of the tubes is so violent that Prof. Bone claims that all scale is thrown off when attaining a thickness of about 1/16 in.

The boiler shown in Fig. 8 has the feed-water heater built on to it, as is the usual construction on the continent. It will be noted that the steam spaces are carefully protected against the heat of the gases. This boiler will evaporate, under normal conditions, 6500 lb. of water per hour. Three combustion tubes are arranged. The length of the boiler tubes is nearly 5 ft. The gases pass these tubes twice. It is plain that in the second nest of tubes there can be no fragments, there being only 13 tubes.

Fig. 9 shows a boiler of very recent construction, with a large superheater. This boiler will normally produce 22,000 lb. of

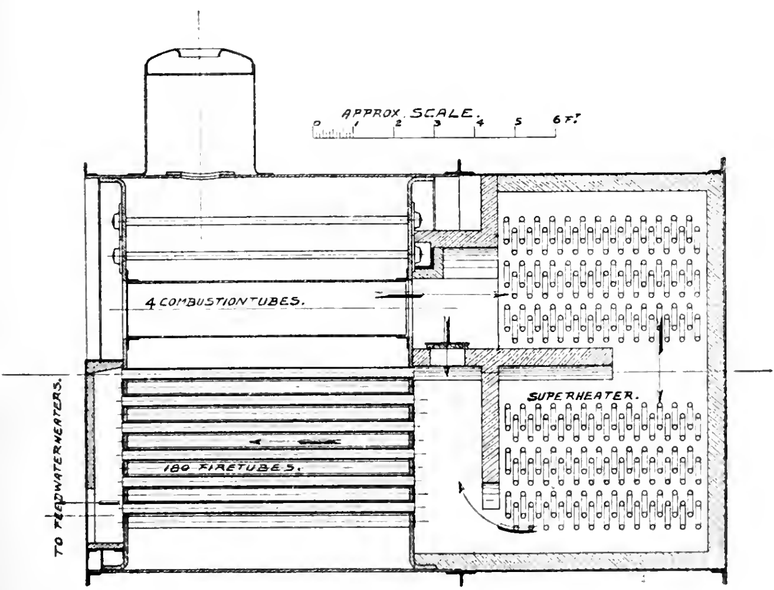


FIG. 9.

steam, and might be rated at 750 h.p. Reference will be made to this boiler later on.

Experiments regarding the use of liquid fuels in combination with surface combustion are said to have been completely successful. The oil is atomized and mixed with air in the same way as with ordinary oil burners, and the mixture is then passed through what is really a muffle furnace (Fig. 10). The combustion occurs partly in this furnace and partly in the combustion tube, which is filled with refractory fragments. The construction shown is very attractive. Two principal disadvantages of earlier designs have been overcome: the gasifying chamber is kept at a very high temperature, which prevents the separating off of

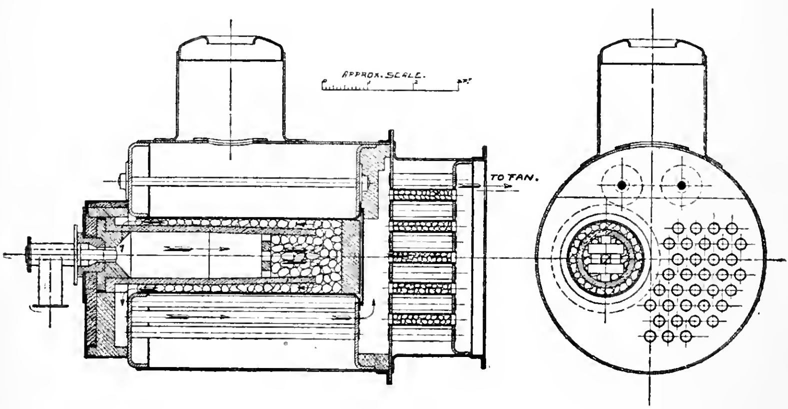


Fig. 10.

free carbon, and no seams are exposed to the first heat. The material of the muffle used must be able to withstand very high temperatures, and no doubt some difficulties will have to be overcome in this respect. The whole problem of the application of surface combustion to practice has been largely one of refractory material.

Boilers of the above description have been made for evaporation of 2600 lb., 3100 lb., and 7300 lb. per hour.

Trials have been made with a small boiler of 130 sq. ft. heating-surface, equipped for burning oil fuel. The results are shown in the table on the following page.

TABLE.

Showing Results of Trials with Small Boiler, Burning Liquid Fuel on the Surface-Combustion Principle.

Date of trial	5/1/1914	7/1/1914	9/1/1914
Duration of trial : hours	8	7	8
Oil Fuel—			
Total used, lb.	1,865	1,809	2,040
Per hour	233	258.4	255
Heating value, B.Th.U., per lb. ..	15,720	15,790	15,970
Water—			
Cold water temperature, °F. ..	50	49.3	48.9
Temperature of boiler feed, °F. ..	200	202	181
Steam pressure, lb. per sq. in. ..	160.5	160.5	160.5
Total evaporation, lb.	22,600	21,620	25,100
Evaporation, lb. per hour ..	2,825	3,089	3,138
From and at 212° F.	2,902	3,180	3,222
Evaporation per sq. ft. per hour	22.5	24.6	24.75
Ratio of evaporation	12.1	12	12.3
,, ,, from and at 212° F.	12.5	12.3	12.6
Waste gases—			
Temperature at exit boiler, °F. ..	716	707	700
Temperature at exit feed heater, °F.	385	293	347
CO_2, percentage	14.3	15.3	16
O_2 ,, ...	4	2.7	1.8
CO ,, ...	0	0.9	0.7
Air pressure—			
At atomizer, inches W.G. ..	34	$31\frac{1}{4}$	$30\frac{1}{4}$
In gasifying chamber, inches W.G.	$20\frac{5}{8}$	$16\frac{3}{4}$	$16\frac{1}{2}$
In feed water heater, inches W.G.	$5\frac{7}{8}$	6	$4\frac{7}{8}$
Power consumption, kw. ..	9.56	8.41	7.77
,, ,, per ton of steam	7.3	5.8	5.3
Recovery of available heat, % ..	90.6	89.2	90.5
Heating surface of boiler, sq. ft. ..	129	—	—

The figures show that the power required for driving the fan amounted to about 6 kw. per ton of steam per hour. Assuming a steam consumption of 20 lb. per kw. hour, this would be about $\frac{20 \times 6}{2240} = 5.35\%$ of the total steam production. However, the fan used was overloaded and belt-driven. It is safe to say that a modern direct-driven set would give a 30 % better efficiency. This would bring the steam consumption of fan down to about 3.75 %. This may be considered the top limit. Careful arrangement of tubes and sizing of fragments will result in a reduction of necessary pressure. and consequently of fan-power. This pressure in recent constructions has been brought down to from 12 in. to 14 in. W.G. with gas-fired boilers. In this case no pressure is needed for the atomizing of the oil, and it appears to be reasonable to put down 2.5 % to 3 % for fan-power.

We can conclude that the maximum attainable recovery of the available heat is—

With oil fuel	..	86 %	With gas fuel	..	91 %

These are no doubt remarkable figures. The reason of the poorer efficiency in the case of oil fuel is partly in the air-excess, which cannot be reduced below a certain point, and is stated to average 11 % in the trials mentioned.

The improvement in heat-recovery from gases through the use of a packing of refractory fragments need not to be confined to ordinary boiler-practice. The use of this means will suggest itself in many instances, but specially where the available temperature difference is small—for instance, in the case of exhaust gases of gas or oil engines. The temperature of these gases may be about 900° to 950° F., and they can be cooled down to 285° F., or even less. No special fan is wanted, the gases being driven through the boiler by the engine itself, and the power required for this purpose is thus reduced to a minimum. A pressure of about 5 in. W.G. would appear to be ample for this purpose, and the power consumption could be put down at about 7 % of the heat-recovery. This high return would be obtained with a very moderate size of boiler, with an evaporation of about 4 lb. of

steam per sq. ft. per hour. The quantity of steam raised would be rather more than 3 lb. per h.p. hour of the oil or gas engine. If we take into account the power used for driving the gases through the boiler, the recovery works out to just about 3 lb. of steam per h.p. hour. This, of course, would be low-pressure steam, at atmospheric or slightly higher pressure, which could be used for heating purposes. If steam of, say, 150 lb. per sq. in. would be wanted, the recovery would be reduced to about 2.3 lb. per h.p. hour. It is important that this saving is possible with a small and inexpensive boiler.

When trying to arrive at an estimate of the advantages and disadvantages of the system which the writer has tried to describe, it should be borne in mind that we are confronted in the majority of cases with the combination of two independent features, viz. :—

(1) Surface combustion.

(2) A packing of refractory fragments for the purpose of improving the heat recovery, or reducing the necessary heating-surface.

The merits and demerits of these two features should be considered separately. The main points in favour of surface combustion are :—

(1) Minimum air-excess.

(2) Very high temperatures attainable without complications.

(3) The heat can be conveniently applied to a well-defined area, and in any direction.

This last advantage is of a special character. In some instances it may be very valuable, but it does not lend itself to a general discussion. While examining the other points, it should be borne in mind that we must compare surface combustion of gas or oil fuel with flame combustion of gas or oil fuel, and not with coal-firing. There is no doubt that ordinary coal-firing is far from perfect, and, if the quantity of fuel burnt is large enough, and markets can be found for the by-products, the manufacture of gas from the solid fuel in a special plant may be profitable. But then, again, the question arises whether surface combustion or flame combustion should be applied.

The air-excess with surface combustion can be reduced to a minimum. If a gas could be produced of absolute uniform properties, this excess might, according to Prof. Bone, be made practically *nil*. But the same is very nearly the case with flame combustion; and, because the combustion is equally perfect in both cases, the products of the combustion will contain exactly the same number of heat units if, during the process of combustion, no heat is given off. It is clear that surface combustion can offer no advantage in this respect.

There is also no reason whatever why, in both cases, the available heat could not be utilized to exactly the same extent, as long as sufficient heating-area in each instance is provided.

As far as thermal efficiency is concerned, there is no advantage on the side of surface combustion as long as sufficient heating-area can be arranged for. However, the high temperatures which are so easily and without complications attainable with surface combustion have a very important bearing on the amount of heating-surface required.

The transmission of heat from a hot gas to a wall is accomplished by radiation and conduction. A formula for the quantity of heat transmitted by radiation was compiled by Messrs. Stephan and Boltzmann. Experiments conducted to find the value of a coefficient for different materials covered temperatures up to 680° F., which, of course, is much too low to arrive at anything like exact figures in our case. Exact figures, however, are not wanted for the argument, and the writer will use the formula referred to only as a convenient expression for the well-known fact that, with increasing temperature difference, the quantity of heat transmitted by radiation increases enormously.

The formula referred to is—

$$R = f\,F\,\frac{T_1^4 - T_2^4}{28.5 \times 10^8}$$

Herein

R = quantity of heat transmitted by radiation in B.Th.U. per hour.

T_1 = absolute temperature of heat-emitting body in ° F.

T_2 = absolute temperature of heat-absorbing body in ° F.

F = area of heating-surface in sq. ft.

f = a coefficient, which was found to be 4.40 for oxidized wrought iron.

In Fig. 11 the values for different temperatures have been plotted out for $T_2 = 1000$, or about 540° F.

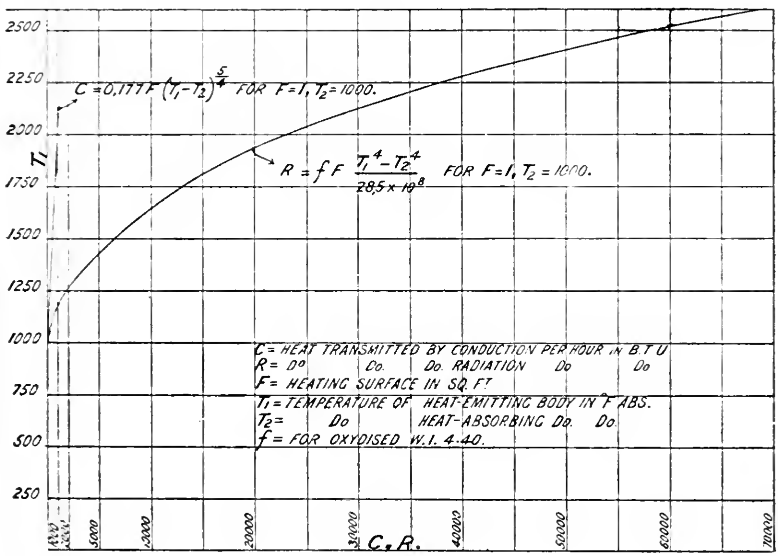


Fig. 11.

A second curve shows the amount of heat transmitted by conduction. According to figures communicated by Prof. Dr. H. Mollier, this heat would amount to—

$$C = 0.177\ F\ (T_1 - T_2)\ \frac{5}{4}$$

The curves, without pretence to correctness, show clearly that we must be able to obtain a considerable reduction in heating-surface if we can increase the average temperature of the heating-gases, which can only be attained without prejudice to the heat-recovery by a high initial temperature. On the other hand, in cases where there is a fixed limited amount of heating-area, a higher temperature of the gases will enable us to force through

this limited surface a much greater percentage of the available heat-units. In such cases, as, for instance, in furnace and muffle work, and with ordinary burners, a considerable saving must be possible in this way. This is fully borne out by tests.

Prof. Bone says * :—

"The most noteworthy point is the relatively low temperature of the escaping products of combustion."

He further states that, comparing a muffle furnace constructed for surface combustion with a good specimen of the ordinary type of gas-fired muffle-furnace, the consumption of gas was found to be as 43 against 105, showing a saving of 59 % on the gas consumption.

In a series of competitive trials made in the United States with American gas-fired furnaces, the surface–combustion system required only half the gas to maintain a given temperature.

Prof. Bone also applied pre-heating of the air, by embedding the air-pipe in a second bed of refractory fragments through which the escaping gases had to pass. This resulted in a saving of from 20 % to 25 % on the already very low gas-consumption, making the total saving against ordinary gas-fired furnaces about 67 % to 69 %. There is, however, no reason why a corresponding additional saving could not be effected with flame combustion.

More instances are quoted in which similar savings are obtained. These are all cases where the amount of heating-surface is limited, and in which surface combustion allows us to force through this limited area a maximum proportion of the available heat.

One should not lose sight of the fact that, in several of these cases, the application of a layer of refractory fragments tends to materially increase the efficiency by mixing and baffling the products of combustion.

The case of a steam boiler will now be considered. In ordinary good practice we are satisfied if the gases enter the chimney at a temperature of about 475° F. This may be regarded as a good compromise. A lower temperature would mean an unduly big and expensive boiler, while a higher temperature would affect the heat-recovery. The benefit to be obtained by the application

* *Engineering*, 1912, p. 633.

of a packing of refractory fragments is very important in these cases, as it will allow the gases to be cooled down to a much lower temperature with a very small heating-surface. We can, by this means, bring the temperature of the waste gases, after leaving the feed-water heater, down to a temperature of about 150° F. above cold-water temperature. This is accomplished with a total heating-surface considerably below that of good modern practice.

The same results, of course, could be obtained without the use of packing, but the size of heating-surface required would be prohibitive. For practical conditions it can be stated that the use of a packing of refractory fragments results in a substantial decrease in heating-surface—*i.e.*, cost and size of boiler—and at the same time in a better thermal efficiency.

This, however, has nothing whatever to do with surface combustion; in fact, there seems to be no reason whatever why, from the point of view of heat-recovery, there is any advantage at all in the application of surface combustion to steam-boiler practice. Surface combustion, of course, would result in an additional, though far less important, saving in heating-area. But it is very doubtful whether this saving is desirable—whether, indeed, it is desirable to utilize the saving due to the baffling of the gases to the full extent.

A reduction in heating-surface means a reduction in the size of boiler, and a reduction also in water-surface, through which the steam generated has to escape. If this surface is made too small, the ebullition will become so violent as to make it impossible to generate dry steam. In several modern boiler constructions this limit has already been reached, and the superheater is relied upon to evaporate what water will be carried over. There is, however, a limit to this undesirable practice, which will result in the blocking of the superheater tubes by incrustations if the boiler water carries even a small amount of salts in solution.

Recent developments in steam-boiler construction are very instructive in this respect. By increasing to the utmost the heating-surface subject to direct radiation from the fire, the necessary total heating-surface is reduced. It is interesting to

note how very carefully the flow of both water and steam is arranged, with a view to make the steam as dry as possible, and to avoid eruptive action in the boiler tubes. This has to be done with an evaporation per sq. ft. only one-third of that attained in the boilers shown. The writer admits that the recent boilers referred to are water-tube boilers. and that with fire-tube boilers (which are used exclusively with surface combustion) the conditions are materially better; but against this is the fact that the heating-surface is required to do three times the duty as compared with modern water-tube boilers.

Those who know how difficult it is to obtain really dry steam in a high-duty boiler will feel inclined to doubt whether the evaporation figures given are not too high, a good deal of what passes through the stop-valve being water.

Now, the recent construction, shown in Fig. 9, is very interesting. The length of the tubes is nearly 7 ft., which is double the length required if the tube were packed with small fragments, and which agrees very well with ordinary marine-boiler practice, and increases very considerably the available water-area. The total heating-surface, without the superheater, is about 1200 sq. ft. The normal steam production is 22,000 lb. per hour, or nearly 18.5 lb. per sq. ft. per hour. This is no doubt a very high figure, but a good deal less than what could be obtained if it was intended to utilize the advantage of a refractory packing to the full. It is significant that the large superheater is arranged so as to be exposed to the first heat, which seems to indicate that the designers expected a high percentage of moisture in the steam even with the lower evaporation figure.

A very small heating-surface and a high heat-recovery are the main advantages obtained in this boiler. It is doubtful whether the first feature will not lead to considerable superheater troubles, and, if this is not the case, whether it would not be far better to increase the size of the boiler so as to be able to decrease the size of superheater, and to expose it to less hot gases. It is important to note that the same advantages can also be attained without the use of surface combustion—namely, by a combination of

flame combustion and a refractory packing of the tube; and so there would be no benefit in the application of surface combustion to steam-boiler practice.

The real advantage to be gained in steam boiler practice accrues from the application of a refractory packing, specially in the case of feed-water heaters.

It is important to note, in this respect, that the velocity of the gases, while passing the fragments, is of vital importance—viz., that the velocity must be high enough to cause a turbulent flow of the gases. The velocity, beyond which parallel flow changes into turbulent flow, is well defined, but can in the instance under review only be found by experiment. A test tube, filled with fragments and surounded by water, could be used, starting with a very low velocity, which could be gradually increased. It would be found that first the exit temperature of the gases, and the total resistance, would gradually rise with the velocity, and that, when this velocity would reach a certain well-defined point, the resistance would suddenly increase, and the exit temperature suddenly drop. At this point turbulent flow would have set in.

It might be thought that parallel flow through a bed of fragments would not be possible, but this is not so. An interesting parallel case, though on a much larger scale, is a blast-furnace stove. Certain experiments, conducted in Neunkirchen a few years ago with Cowper stoves, led to the conclusion that in ordinary practice the velocity of the gases when heating up was not high enough to create turbulent flow. The velocity was increased by means of an air-jet. The exit temperature of the heating-gases rose with the velocity to a point at which the rate of flow was about 60 % above normal. A further increase in velocity resulted in a lowering of the exit temperature, turbulent flow saving set in, the heat-recovery increasing rapidly with the rate of flow. Several steel works in that district have now adopted the high-velocity Cowper stove, and only two stoves are now wanted for one blast furnace; or, if the four existing stoves are run on the new principle, a good saving in fuel results.*

* *Engineering*, 1914, p. 394.

It may be noted, however, that this is not the first attempt to destroy the effect of cold layers of gas, which act as an insulating medium between the boiler wall and the hot gases. Several attempts have been made to mix the gases by the application of jets of steam or by mechanical appliances. These attempts were, however, generally limited to the zone over and just behind the fire-bridge, and aimed in the first instance at an improvement in the process of combustion. As far as the writer is aware, none of these attempts have been a signal success, except in cases where the combustion was very bad.

It might be asked why the heat-recovery in the ordinary type of gas-fired boiler is often so low. In many cases this recovery does not reach 60 %, while 70 % is regarded as an excellent figure. The reason seems to be mainly in the dust which is carried over by the gas and deposited on the heating-surface, where it acts as an insulating layer. In addition, even when the boiler is quite clean, the temperature of the waste gases is much higher than would be the case if baffling had been arranged.

The tendency is to pay increasing attention to the cleaning of the gas, and, from what has appeared in the press, it seems to be fair to conclude that the gas, even if it is used for boiler-firing, will repay cleaning to such an extent as to be also good enough for surface combustion, and for the use of a packing of refractory fragments. In this respect the new system would not be at a disadvantage.

In some industries where combustible gases of a low calorific value are available, these gases could be utilized in processes where high temperatures are required; but these temperatures cannot be reached without regeneration, making the plant expensive and complicated. In such cases surface combustion would be a means of attaining these high temperatures in a very simple and inexpensive way, and might reduce first cost and cost of upkeep sufficiently to make the utilization of the poor gas profitable. It is in cases like these, and in such where the amount of heating-surface is limited, that the application of surface combustion may result in considerable saving.

Papers and Discussions.

CONTENTS.

AN EXAMPLE OF LOW WORKING COSTS.

BY V. F. STANLEY LOW.

THE following brief paper is submitted with a view to encouraging the exploitation of hitherto neglected low-grade ore-bodies which are suitable for extraction by open-cut methods.

The Corinthian North mine, situated 10 miles from Southern Cross, in the Yilgarn district of Western Australia, furnishes an example of low working costs, although operations are carried out on quite a limited scale.

The body of quartz which is now being worked strikes N.W. and S.E., has a very slight underlie to the west, and lies between bodies of quartz-hæmatite schist on the east and amphibolite on the west.

So far as the property has been explored, it would appear that the underlying sulphide rises to within about 70 ft. of the surface in the northern end of the workings, and pitches away to the south at a grade of about 7½%.

Only oxidized ore is being mined at the present time. The sulphide takes the form of pyrrhotite, and it will therefore be

[This paper is separately bound, and may be, if so desired, detached complete from this number.]

necessary to dry-crush and roast the ore from the lower levels before submitting it to cyanidation.

The oxidized ore averages roughly 20 ft. in width, and the depth available for removal above the 100-ft. level throughout the length covered by present extraction operations approximates 90 ft.

Communication between the mine and seaboard is satisfactory, as the railway from Southern Cross to Bullfinch traverses the leases; and a good supply of fresh water is furnished by a branch line of the Western Goldfields Water Supply.

The surrounding country is covered with timber which is excellent alike for fuel and mining purposes.

The main shaft has been sunk in the footwall to a depth of 210 ft., and cross-cuts have been put out from it to intersect the ore at depths of 100 ft. and 200 ft. From these cross-cuts drives have been carried north and south within the ore-body, the drives at the 200-ft. level having been put in for the purpose of exploring and opening-up the sulphide zone. The drives at the 100-ft. level are immediately under the open-cuts, with which they are connected at suitable intervals by means of ore-passes. It is through these passes and drives that all broken ore eventually reaches the main shaft and stamp battery.

Owing to the proximity of the main shaft to the open-cuts, and of the fitting shops to the main shaft, the surface is somewhat cramped for room; but otherwise the plant is compact, and eminently suitable for economical working.

The steam plant consists of Babcock boilers adapted for wood fuel, air compressor, winding engine, mill engine, and electric-light engine. Steam is generated at 150 lb. working pressure, and approximately 186 tons of firewood are consumed weekly. Fouché condensers help to economize in fresh water supplies for boiler purposes, and an adequate supply of salt water for battery and cyanide purposes is pumped from the underground workings.

The treatment plant consists of an ore-storage bin, 20 head of stamps, 2 tube mills, 2 cone classifiers, 2 centrifugal pumps, 2 thickeners, 4 agitators, 2 Ridgway filters, 2 double-ram pumps,

salt-water tank, battery tank, 3 precipitating boxes, and the necessary furnaces for roasting and smelting. In addition to the above, there are 2 grinding pans installed, but the use of these has been discontinued under present management.

The broken ore is delivered from the ore-passes into trucks at the 100-ft. level, is run to the plat, and raised through the main shaft to the brace, 48 ft. above surface level. From the brace the truck goes to the tippler, whence the ore passes over a grizzly to a 22 in. x 11 in. crusher of the Blake type. The fine from the grizzly and the crushed ore go direct to a rectangular bin 34 ft. x 14 ft. x 18 ft. 6 in., from which they are automatically fed to the 20-stamp battery.

Each of the stamps weighs 1250 lb., has a drop of 8 in., and drops 106 times per minute. The depth of discharge is set at 1½ in., and the order of drop of the stamps is 1, 3, 5, 2, 4. The duty averages slightly over 10 tons per stamp per 24 hours. Screens of 17 and 16 gauge wire with 6 and 10 openings to the lineal inch are used, according to the nature of the ore to be crushed; the larger mesh of the two is the more generally used.

As a rule, the material leaving the battery has the following approximate sizing:—

	%
On 20 mesh	34
„ 90 „	39
„ 150 „	5
Through 150 „	22

It flows through launders to a 4-in. centrifugal pump working at 650 r.p.m., by which it is delivered, through a head of 27 ft., to two cone separators, 6 ft. in diam. and 7 ft. 6 in. deep. The overflow from these cones has an average grading of—

	%
On 90 mesh	25
„ 150 „	19
Through 150 „	56

and enters two circular thickeners 25 ft. in diam., the arms of which make 30 revolutions per hour.

The underflow from the cones has an average grading of—

	%
On 40 mesh	31
,, 90 ,, ...	55
,, 150 ,, ...	6
Through 150 ,, ...	8

and flows through launders to two 16 ft. x 4 ft. 1 in. tube mills, which make $31\frac{1}{4}$ r.p.m.

The discharge from the tube mills has an average grading of—

	%
On 40 mesh	13
,, 90 ,, ...	60
,, 150 ,, ...	10
Through 150 ,, ...	17

and contains approximately 55 % of solids; it joins the battery discharge, flows to the centrifugal pump and delivers the whole to the cone separators.

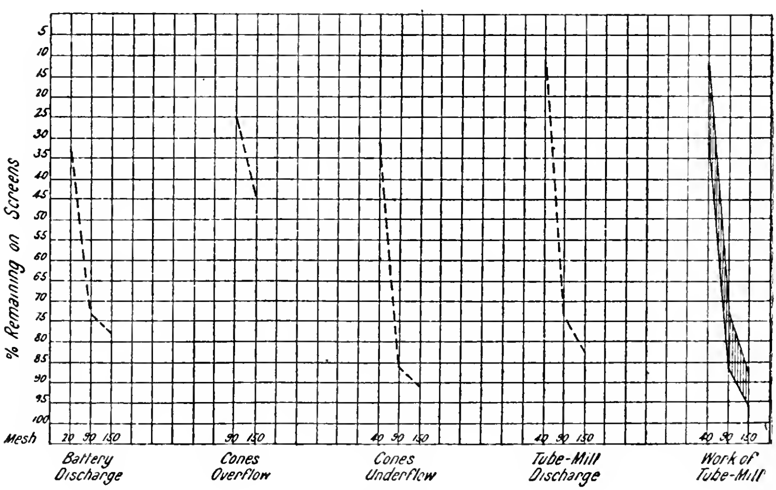
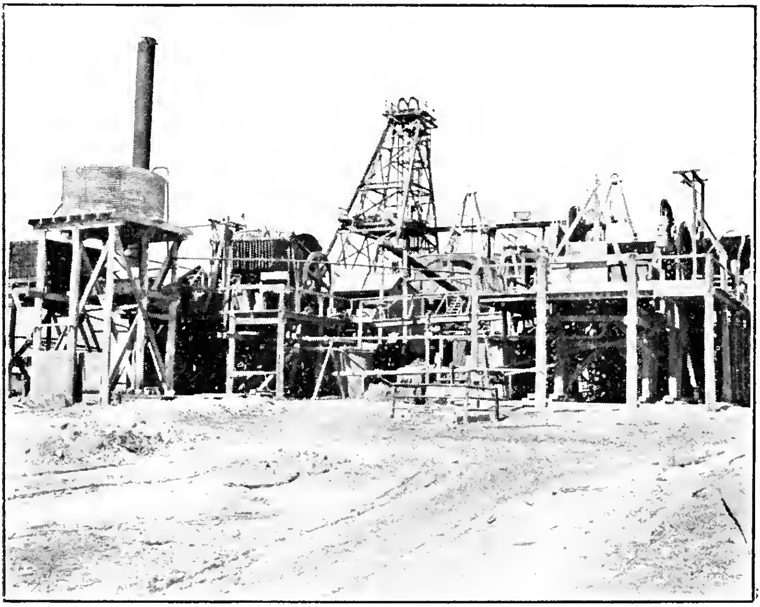


Fig. 1.
Sizing Diagram.

The underflow from the 25-ft. thickeners is delivered to four agitators, each 18 ft. in diam. and 8 ft. deep, the arms of which

Open Cuts looking North, Corinthian North Mine.



Portion of Plant showing Ridgway Filters, Corinthian North Mine.

make 15 r.p.m. From each of these agitators in turn the pulp is pumped to the Ridgway filters, each containing 20 filter leaves, which measure 5 ft. 6 in. x 4 ft. 2 in. each. A vacuum of 20 in. is maintained on the filter leaves during caking and washing operations. The washed residue is pumped away to the residue dump, the grading of the residue corresponding very closely with that of the overflow from the cone separators.

Cyanide is added to the pulp as it enters the agitators, and the wash water from the filter plant is pumped to the battery tank. Thus the ore is brought into contact with cyanide solution as soon as it enters the battery; and no mercury is used in the milling operations except for the amalgamation of the slags formed in smelting for bullion.

At the present time the total recovery of gold averages 90 %.

The total number of shifts worked daily in all departments on the surface and underground averages 76, and the following is the schedule rate of wages paid :—

Rates of Wages.

	Per shift.		Per shift.
Mill men	15/-	Carpenters	15/-
Filter men	13/4	Riggers	11/8
Engine drivers	14/-	Horse drivers	11/8
Firemen	11/6	Labourers	10/6
Trimmers	10/6	Miners	11/8–14/4
Fitters	15/-	Truckers	11/-
Blacksmiths	16/8		

Driving, cross-cutting, sinking, and rising are done on contract at prices varying in accordance with the nature of the work to be performed. Machine drilling in the open-cuts is contracted for at so much per foot drilled; but the spalling and rilling of ore to the chutes is done on wages.

Trucking from the chute to the plat and from the brace to the cracker is contracted for at 10d. per ton; and, similarly, mullock is delivered from the chute to the dump at a contract price of 1s. per ton.

The consumption and cost of stores for the month of January is given in the following table:—

CONSUMPTION AND COST OF STORES.

JANUARY, 1915.

Description.	Cost.	Expenditure. £	s.	d.
Candles	6.58d. per lb.	7	19	0
Gelignite	9.66d. per lb.	66	16	6
Fuse	6.75d. per coil	2	14	0
Caps	3s. 4d. per box	0	16	9
Cyanide	10.9d. per lb.	133	17	9
Zinc	6.9d. per lb.	30	11	1
Lime	2s. 9.5d. per bag	76	17	3
Filter cloths	18s. 3.3d. each	18	5	6
Acid, Sulphuric	7s. 1.7d. per jar	3	3	0
,, Nitric	19s. 3.6d. per jar	5	7	0
Borax	6d. per lb.	5	12	0
Flints	£4 12s. 8d. per ton	77	17	8
Kerosene	5s. 10d. per tin	0	17	6
Firewood	6s. 9d. per ton	254	11	2
Water	10s. per 1000 gal.	99	0	0
Battery shoes	£1 3s. 3d. per cwt.	26	4	4
,, dies	£1 3s. 3d. per cwt.	11	5	0
Wire screen	£7 10s. per roll	3	15	0
Coke	£4 15s. per ton	9	10	0
Blacksmith's coal	4s. per bag	6	17	1
Oils	..	30	14	0
Steel	..	9	7	6
Iron	..	2	10	0
Pipes and fittings	..	11	7	9
Bolts, nuts and nails	..	4	4	5
Rock-drill parts	..	2	0	5
Packings	..	21	15	4
Grease	..	4	8	1
Waste	..	3	0	9
Cement	..	4	1	0
Belting	..	27	2	2

NOTE.—The consumption of cyanide, zinc, lime, packings, filter cloths, grease, waste, nitric acid, and flints was above normal; that of candles, fuse, caps, and borax was slightly below normal.

The costs incurred in breaking, raising, and treating 5230 tons of crude ore and in carting and treating 418 tons of dump tailing during the month of January were:—

Breaking and delivering 5230 tons of crude ore—

	Per ton. s.	d.
Superintendence ..	0	1.19
Contractors' earnings	0	11.06
General labour ..	0	9.01
Hoisting	0	2.44
Explosives and candles	0	3.11
General stores ..	0	2.43
Rock-drill expenses ..	0	2.52
Electric light ..	0	0.05
Total ..	2	7.81

Treating 5230 tons of crude ore and 418 tons of dump tailing—

	Per ton. s.	d.
Superintendence ..	0	2.28
Assaying and sampling	0	2.24
General power ..	1	2.34
Pumping mine water	0	1.35
Electric light ..	0	0.39
Spare parts	0	4.01
General repairs ..	0	8.10
,, stores ..	0	2.78
,, labour ..	0	10.01
Flints	0	3.32
Zinc	0	1.30
Cyanide	0	5.69
Lime	0	3.26
Carting tailing ..	0	0.76
Clean-up	0	1.60
Air compressor ..	0	2.00
Total ..	5	3.43

SUMMARY.

5230 tons of crude ore—

	£	s.	d.		Per ton. s.	d.
Ore breaking and transport	693	3	7	..	2	7.81
Removal of overburden	158	10	1	..	0	7.27
Treatment	1382	6	7	..	5	3.43
Realization	57	10	5	..	0	2.64
Management and general expenses ..	274	2	6	..	1	0.58
Total	2565	13	2	..	9	9.73

418 tons of dump tailing—

	£	s.	d.		Per ton. s.	d.
Treatment	110	9	6	..	5	3.43
Realization	4	12	0	..	0	2.64
Management and general expenses ..	21	18	2	..	1	0.58
Total	136	19	8	..	6	6.65

Total working expenditure, £2702 12s. 10d.

In addition to the foregoing, £123 15s. 6d. was expended in developmental work, being equal to 5.58d. per ton of ore raised.

Management and general expenses includes the salary of the General Manager.

Taken over a period of 12 months, the average cost of developmental work has been as follows :—

	s.	d.
Sinking ore-passes	38	1
Driving	46	6
Cross-cutting	47	1
Rising	29	3

The cost of rising appears to be low when compared with that of the other branches of developmental work ; but it is the practice to carry individual rises only a very few feet above the levels.

The working costs for January show a considerable reduction over those for the year 1914, which were as follows :—

	s.	d.
Ore extraction	3	5.83
,, treatment	5	10.06
Management and general expenses	1	9.13
Total	11	1.02

There is every reason to believe that costs will be kept in the neighbourhood of those quoted for January, so long as the present system of work can be maintained ; but, even basing calculations on the working costs obtained during the full year of 1914—viz., 11s. 1.02d. per ton—and making due allowance for interest on capital, depreciation, &c., it will be seen that it does not require a very rich ore-body to pay a profit in such cases as that just described where it is possible to mine on the open-cut system and obtain a good gold-recovery by direct cyanidation.

MINING EDUCATION IN AUSTRALASIA.

BY D. B. WATERS.

Discussion.

MR. H. W. GARTRELL wrote—the somewhat complex arrangement between the Adelaide University and the South Australian School of Mines seemed to have misled Professor Waters. On page 4 he stated that the University did not itself provide instruction in the technical subjects relating to mining; on page 5 he made no mention of any degrees being granted, and on page 7 he spoke of the New Zealand students having to take two sets of papers and gave the impression that Adelaide students had to do likewise. The facts were that the University of Adelaide had a lecturer on Mining Engineering; that it gave the degrees of Bachelor and of Master of Engineering, and that there was no duplication of papers. The position differed from that of other Australian mining schools in that when the University began to give instruction in mining there was situated next door a successful School of Mines giving three-year Associate diplomas in mining and in metallurgy. An agreement was made to create a new four-year course in mining and metallurgy—since divided into separate courses—to be given jointly and consequently carrying two diplomas, the Fellowship of the School of Mines and the diploma in applied science of the University. To those students who fulfilled the University matriculation requirements the degree of B.Sc. was given.

Two years ago, in order to fall in line with the other Australian Universities the degree was changed to Bachelor of Engineering, and provision was also made to grant the degree of Master of Engineering. The lecture courses in mining number four—two in mining engineering and two in ore-dressing. The ore-dressing was taught at the School of Mines but all the classes were under the University lecturer in mining engineering. The four-year

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mining men took all four courses; the four-year metallurgy men two courses in ore-dressing and one in mining engineering; the three-year mining men took one course in mining engineering and one in ore-dressing; and the three-year metallurgy men one course in ore-dressing. There was thus every facility for a student wishing to take more than one diploma.

Referring again to the paper they found, on page 16, the suggestion that diploma and degree examination papers were usually set by theoretical, non-practical men. He (Mr. Gartrell) did not know of any important Australian institution whose mining papers were so set. It was to be expected that teaching and practising engineers possessed theoretical and practical knowledge in different proportions, but surely it was as unfair to call the one untheoretical as the other unpractical. The mining papers at Adelaide were set and examined jointly by the lecturer and a prominent practising engineer—a very satisfactory arrangement. It was certainly desirable that there should be substantial uniformity in the titles and standards of degrees, but was the present state of affairs so very bad? In all the mainland States the degree course in mining was four years and, with the exception of Victoria, the degree was B.E. In Victoria it was B.M.E.

The important Schools of Mines, with the exception of Western Australia, gave three-year courses leading to an associate diploma. In Kalgoorlie that diploma was given for a course nominally four years but about on a level with the others. It would be a distinct gain if all these schools gave the "degree" A.S.M., and he thought proper representations would readily convince the authorities. There would then remain but three "anomalies," the extra letter in the Melbourne degree, the title of the Melbourne diploma, and the Fellowship of the South Australian School of Mines. They had thus in Australia three degrees, Master and Bachelor of Engineering, and Associate of School of Mines, and the standards varied very little in different institutions. Professor Waters appeared to wish to abolish the first two and to make a new degree having requirements intermediate between those of the Bachelor's and Master's degrees, and then to give it the title

Associate of School of Mines. The reason given was that that was probably the best known designation in the mining world. That of course was quite incapable of proof or disproof, but if one were to make a list of all the mining schools giving a diploma up to the degree standard of, say, Adelaide or Melbourne, the writer thought it would be found that the commonest distinction in English speaking schools would be B.Sc., in British schools B.E., and that Associate of School of Mines would occur about twice. No doubt A.S.M. was the most commonly held diploma in Australia—Adelaide had over a hundred Associates in mining alone—but that seemed to him a strong reason for it to continue its old significance.

The "sandwich" system of education advocated was one that needed careful examination. It was being tried at the University of Western Australia, which published a note in its calendar saying that the system had been approved in a general way by a large number of English civil engineers. It was indeed in successful use in U.S.A., notably at the University of Cincinatti, but it should be noted—

1. The system was rare, even in U.S.A.
2. The University of Cincinatti, though containing an important engineering school, was far from being a typical American University.
3. The particular scheme advocated appeared to miss the main advantage of the Cincinatti scheme, and to have all its disadvantages and more.

The objections to the system fell under two main heads—the technical difficulties of carrying it out, and the ruin of educational ideals involved.

A very considerable part of a course in mining consisted of pure and applied mathematics, physics, chemistry, geology, and mineralogy. Those subjects were also taken by students in arts and pure science, and if they were to be taken in an acadamic year of six months the classes must be duplicated, a most wasteful method at any except a large University, or the Faculty of Pure Science must arrange its work to suit the Faculty of

Engineering. Seeing that the ordinary academic year was about nine months they might then anticipate confusion and ineffectiveness to mark the work in pure science. But supposing the student to have received his six months' instruction under the most favourable conditions and to proceed for his practical work to a camp where there was a good mining school. They could not expect that school to work all the year, and presumably the student arrived in time to begin the third term's work. Was he to have special short classes started for him? And would he be so free from trouble in the first month that evening classes would appeal to him? There was no need to labour the difficulties of a mining course on those lines, they would be great for a large rich University and would increase rapidly for the smaller and poorer ones.

Without going into details of the Cincinatti experiment it should be sufficient to say that the increased practical knowledge obtained was a quite secondary object. The success was greatest in the mechanical school, and while he had no doubt that its organiser could, though with much greater difficulty, make a success of a mining school on similar lines, it must be remembered that it would be a success under American conditions, with American students and from an American point of view, and those were not wholly Australasian. It was this he (Mr. Gartrell) had in mind in speaking of the ruin of educational ideals, for in those they were, fortunately, far more British than American, though they had much to learn from America about efficiency in instruction.

The chief advantage of learning mining at a University instead of at a mining-field school lay not in the better equipment, or more specialised instructors, or in the greater time of instruction; it lay in sharing the full academic life with men of other schools and professions. Start a "sandwich" system and the engineering men were a class apart. Even if numerous enough to do many things together they were being robbed of a most valuable part of their education. In almost all Australian institutions a "sandwich" system in mining would be a method of technical

instruction inferior to the present method. In all it would be a failure as a method of technical education—an allied but much bigger matter. They could not, however, escape the fact that practical knowledge made lectures much more profitable. The student had three long vacations in which he could get a good deal; although part of two of these should be devoted to survey camps. He (Mr. Gartrell) had for some time been of the opinion that the best results were to be obtained by spending the fourth year in the field and then returning for the final year's lectures. The student had acquired a good elementary knowledge of his professional work, he could, if he wished, go to a field too distant for a summer excursion, and he could get in a goodly portion of the work at the face, required for his mine manager's ticket. In many cases the money earned would be acceptable, and finally, when graduated, he could set out much more confidently in search of employment.

Professor Waters accused the mining schools of not "recognizing their duty, in respect to practical experience requirements, to their students." This could he (Mr. Gartrell) thought be readily refuted on two grounds. A large proportion of the men taking mining degrees did not remain permanently in the industry, and one was almost tempted to say that the better the school the larger this proportion; moreover that state of affairs was not peculiar to Australia. The fact was that a good mining course was an unequalled general training in science, and the young graduate did not find it difficult to "make good" in almost any branch of engineering. That unusually good opportunities in other fields of work, or dull times in mining, took many of their graduates into industries other than mining was not to be lamented so long as those men found their training well suited to their new work. But were such men to be denied their degree unless they remained three or four years in practice instead of the one year at present required? Further, he could not admit that a first degree should include everything necessary for a final State qualification as a mine manager. It seemed to him that before obtaining such a qualification a man should, for a con-

siderable time, get right away from his instructors. The better the teacher the more would he colour the views of his students, but to be sure on his feet a man must develop his own view point, and he could do this very much better in two or three years' consecutive practice than in the same amount of time taken as half-yearly sandwiches.

Finally he would say that while he had tried to show that the mining schools did not deserve all the severe handling they had received from Professor Waters, he was well aware they did deserve some of it, and that they would greatly benefit if the Institute would seriously take up the discussion of the five heads enumerated by him.

Professor E. W. Skeats wrote that Professor Waters' interesting paper on Mining Education in Australasia raised and discussed a number of the important phases of the subject. From among those one or two points suggested themselves for discussion. Professor Waters recognized two main types of schools—the Universities and certain of the larger Schools of Mines constituting one class while smaller and local Schools of Mines comprised the other. In the writer's opinion, at least three classes were represented. Speaking generally the University schools gave rigorous practical and theoretical courses based on a relatively high standard of preliminary education as tested by a compulsory matriculation examination. Certain of the larger non-University Schools of Mines gave admirable courses, especially in the practical and technical subjects; while the theoretical side of the training, the training in the fundamental sciences, and the requirements of preliminary general education, were decidedly of a lower standard than in the University schools. The smaller and local Schools of Mines were usually less well equipped and gave a training of more restricted scope and mainly of a technical character. It was clear then that the smaller institutions provided training mainly for the subordinate posts; the larger institutions mainly for the more responsible posts in the mining profession. Of course

it was recognized that some students from the smaller institutions, in spite of lesser preliminary education and less exhaustive technical equipment when possessed of the requisite brains, energy and business capacity had reached and would continue to reach high positions in the profession, while some highly trained mining students in spite of greater advantages from a variety of causes might and did fail to pass beyond the subordinate positions in mining.

Professor Waters advocated the "sandwich" method of training, namely six months in the mining school and six months in the mine, for each of four years. That system obtained only in Dunedin and was only practicable in Dunedin among Australasian schools. It was adopted there because, as a survival of the Scottish system, the academic year in Otago was only of six months' duration. In all other University mining schools in Australasia the academic year was of nine months' duration, and in most of the non-University mining schools reached ten months. There was no doubt something to be said in favour of the Dunedin practice, but it must be remembered that in comparing a nominal four years' course at Dunedin with a nominal four years' course at an Australian University the total actual periods of University training were really two years at Dunedin and three years at an Australian University. If such a practice were feasible and adopted in Australia its most obvious result would be the sacrifice of one whole year of theoretical and practical University training in subjects important to the professional equipment of the young mining engineer. Personally he (Professor Skeats) preferred the Melbourne practice of requiring short periods of practical work in mines in vacations during the University course, and demanding a full year's experience after the course was completed, before granting the degree of Bachelor of Mining Engineering. He did not attach the same importance which Professor Waters did, to uniformity in the title of the degree or associateship which was granted to the student on completing his course. No doubt if they were all starting afresh in mining education in Australasia it would

be convenient to award a degree or diploma of similar nomenclature to courses of a comparable character. The really essential matter however was to see that the course was thoroughly and effectively directed towards the higher technical and professional equipment of the students. The student's school or University could only provide him with the groundwork or foundation of his professional career and if the teaching were thorough and the course wisely mapped out would give him the mental breadth and outlook which would, with experience, enable him to rise to positions of responsibility.

With regard to the vexed question of how long a period of practical work in mining should be demanded and what the nature of the experience required should be before the young mining engineer was granted the mine manager's certificate he held the following views: after completing a mining course at the University or one of the larger Schools of Mines no further examination should be required. Three years' practical experience should be sufficient for such a trained man, of which twelve months' manual work underground especially in timbering and drilling should be required. Very little time should be required in practice at unskilled work, such as trucking, and the remainder of the period of practical service could well be done while holding a subordinate position in assaying, underground surveying, &c. The manual work, especially the unskilled work underground, might be done, in part at least, in vacations during the course of training. No real hardship to the young mining engineer would accrue from such a period of apprenticeship, as it was highly improbable that he would be given the post of manager of a mine with less than three years' practical experience. On the other hand the period of service for the mine manager's certificate should not, he thought, be protracted beyond three years of diversified service. The practice of some of the Australian States of demanding two to three years experience at relatively unskilled work underground at the face put an unreasonable handicap on the young professionally trained mining engineer.

Papers and Discussions.

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PRESIDENTIAL ADDRESS.*

PYRITE SMELTING AT MOUNT LYELL.

BY ROBERT C. STICHT.

I WISH to express to the members of the Institute my sincere appreciation of the distinction which they have conferred upon me by again making me their president. It also gives me great pleasure to once more welcome the Institute to Mount Lyell. Since your last visit certain improvements in plant have been made, which I trust you will have found interesting, and which show that, though located in a remote corner of the world, we are abreast of the times.

The opportunities which a recent lengthy trip to America has given me to observe the degree of interest which our local smelting practice continues to arouse in metallurgical circles make it seem appropriate that I should take advantage of your visit to lay before you some detailed information on the more practical features of pyrite smelting, particularly as carried out at Mount Lyell. There are only a few places in the world where the method is now being practised in approximate purity—only about half

* First Ordinary Meeting, 1915, Queenstown, Tas.

[This paper is separately bound, and may be, if so desired, detached complete from this number.]

a dozen, far apart—Tasmania, Tennessee, Rio Tinto, Algiers, Japan, Russia. It is, therefore, of rather restricted usefulness, yet its striking departure from former beaten tracks, and its special scientific interest, have caused an extensive literature to grow up around it, and there is no doubt that its underlying principles have important bearings on other more common branches of metallurgy. However, it is a kind of work in which doing is decidedly more difficult than speculating on "underlying principles," and but little has so far been published on the manipulative side of the work at Mount Lyell. Pyrite smelting has here been carried on, without interruption, since 1896, and first came into prominence here, and Mount Lyell is perhaps still the foremost instance of its application. It became obvious to me, from the eager curiosity displayed by my American colleagues, that the points on which information was most desired were those turning about "how it is done," and not so much "why it works." I propose, therefore, to dwell principally on the former aspect of the method, though it is necessary, for a satisfactory presentation and a clear understanding, also to touch upon the feature of "how it works," while we can leave the "why" in the background.

I will spare you the description of the apparatus, since this does not differ so very greatly from that used in ordinary copper blast-furnace smelting. The two accompanying figures will make the construction of the furnace sufficiently clear for the purpose in hand, even though they are now a little out of date.

The general chemical principles of the method are also matters of current knowledge, but it will be well to dwell a little on certain physical or mechanical features, for it is on these, as a foundation, that our practice rests. Where the practice is more difficult than ordinary copper smelting in blast furnaces, or seems to flatly contradict certain tenets of the latter, the cause is to be found in the features referred to.

The essential characteristics of the treatment have always been the same, but the practice of to-day differs in a few minor respects from that of the past. the departures being primarily owing to

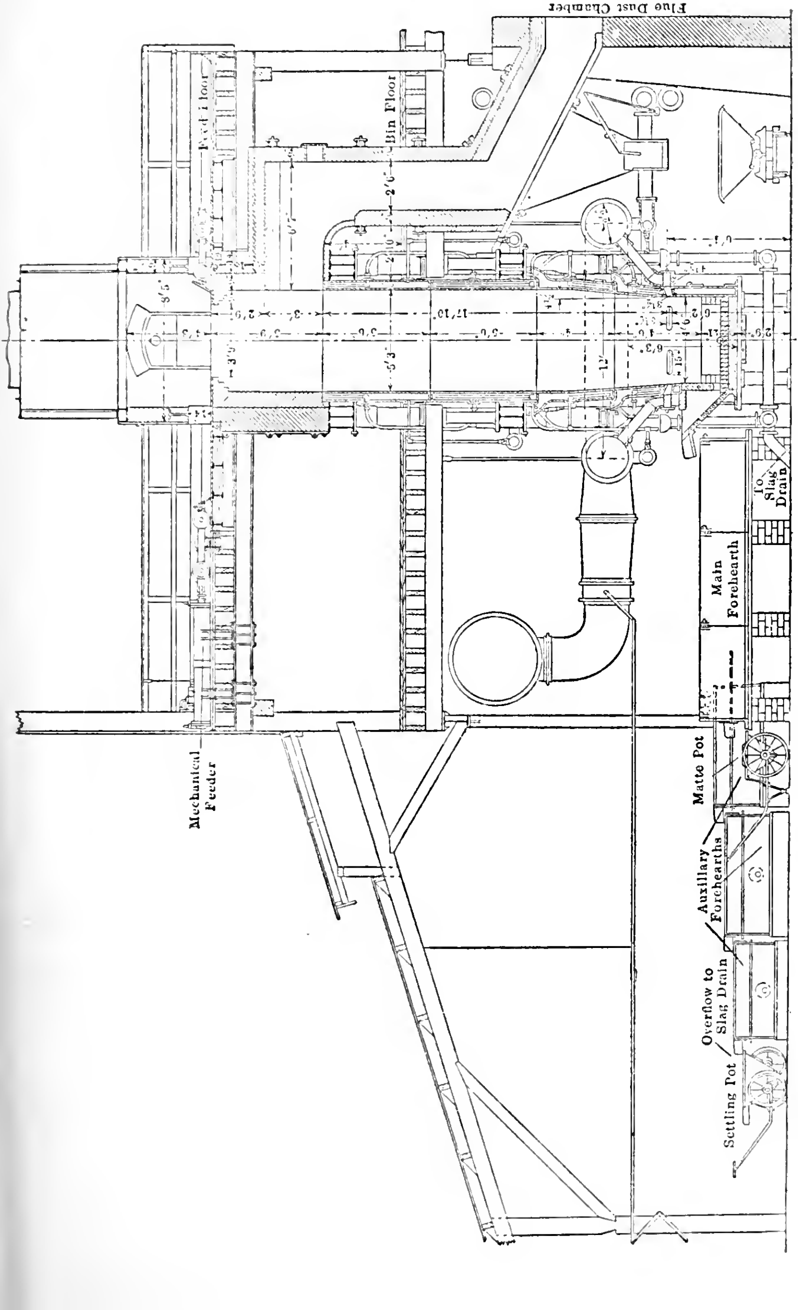


FIG. 1.
MOUNT LYELL BLAST FURNACE—CROSS SECTION.

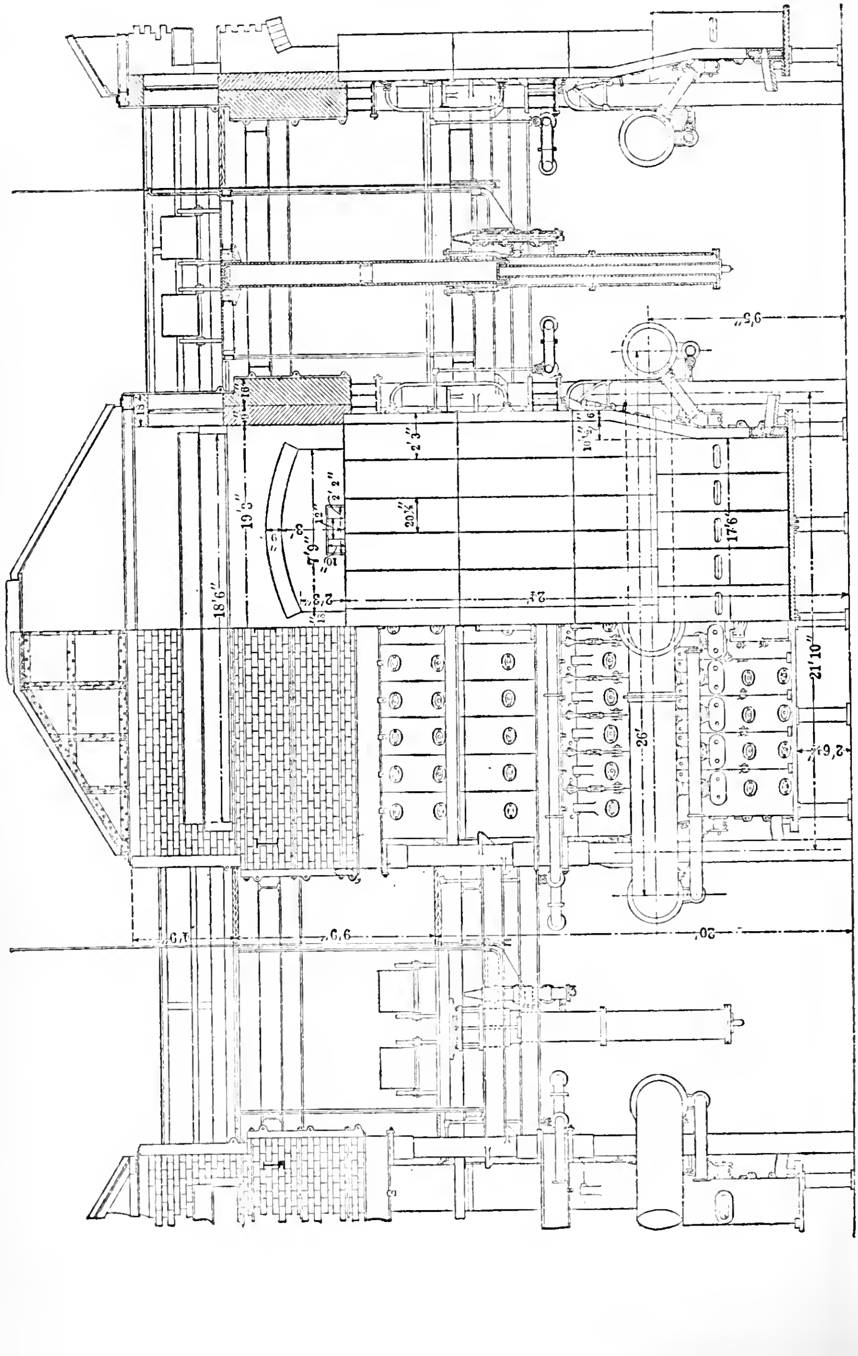


FIG. 2.
MOUNT LYELL BLAST FURNACE—LONGITUDINAL SECTION.

slight changes in the composition of the ores which have taken place incidental to the attainment of greater depth in the mines. It would lead too far to deal in detail with any other period of our metallurgical history than the present, and so the experiences and lessons of the past, including such interesting subjects as that of a heated blast and the current use of extremely low coke percentages (0.1 % per half-year, etc.), must be left almost untouched.

In the first place, then, the treatment, as now carried out, is, in essence, a continuous bessemerizing of massive sulphide ores of iron, with more or less copper, in a shaft-furnace, together with highly silicious ores, also containing more or less copper, up to the stage of a high-grade matte, in one smelting. The ores are wholly devoid of oxides of the metals. The fuels used are the two chemical constituents of iron pyrites—viz., iron and sulphur—and a modicum of coke. A considerable volume of blast is applied, with a great pressure for shaft-furnace work in the non-ferrous metals. The use of hot blast, without which the process could not be made to work in its earliest days, has long ago been discarded. It is no more necessary in our work than in the converting of mattes into blister copper. Its advantages have been replaced by a much higher furnace activity, which is achieved by means of a greater volume of blast, with, incidentally, a higher pressure, and it has also been necessary to slightly raise the coke consumption.

Mount Lyell started with 18 oz. of blast pressure, and now sometimes uses 96 oz., to the sq. in. The quantity of free air blown into a furnace is a minimum of 1000 cub. ft. per foot of hearth length. The converting of mattes is possible with only 8 lb. of wind. The distinction between the action of the converter vessel and the shaft furnace is thus disappearing in our case, as far as mere blast *pressure* is concerned. The distinction between the apparatus will, however, continue to maintain itself, as far as blast *volume* is concerned, owing to the difference in the quantity of iron and sulphur to be oxidized and scorified in comparison with the copper to be saved.

Our discarding of hot blast has not, however, settled the point whether the use of a heated blast would not, metallurgically, improve our pyrite smelting even now. The introduction of extra heat through the blast, instead of its derivation from an increment of coke, ought *a priori* to be an advantage under all circumstances, and we have no experience with hot blast under our present high pressure, high volume conditions. The fact that a different slag would form could be dealt with successfully. But the greatest drawback of a heated blast is its cost, and there is no hope of overcoming this defect. It greatly exceeds the cost of the additional coke and of the greater mechanical power now required for the supply of the blast.

In order to understand our method, it is necessary to form a correct mental picture of the inner appearance and working condition of the pyrite furnace. Its most salient feature is a pronounced openness or porosity of the masses in action, together with a total lack of stratification or bedding, yet with an even distribution of the various components of the charge throughout each horizon. There is no layer-like arrangement of the different materials, and no alternation of charge and fuel. There is no reducing zone, but an oxidizing one, restricted, like the first-mentioned, to the lower regions of the column. In distinction from the ordinary furnace, in which a distinct reducing zone precedes the slag-forming zone, the oxidizing and the slag-forming zone are identical. From top to bottom, however, there is one element of the charge present everywhere, which forms the skeleton, so to say, of the furnace-body in action. This element is the *free silica* in charge. The lumps of quartzite, etc., are present in excess of the quantity required for the momentary furnace action, and they build up a honeycombed or vesicular column, reaching from the crucible of the furnace upward to the stock-line, or top of column. This silica skeleton is a necessity. Without it pyrite smelting is unthinkable. It forms a physical structure like an aggregate of cells, each cell in the heart of the furnace forming a little bessemer vessel with silicious walls. The wasting away of the material of the walls with scorification does not unsettle the fixity of the column, for the lumps of silica

are constantly being replenished by new ones settling down from above.

More particularly, the furnace column may be divided into three regions—an upper preparatory one; a central one, where the entire chemical activity takes place; and a lower one, extending from above the tuyeres to the bottom of crucible, in which no action takes place. The first contains the charge undergoing only primary chemical and physical changes, the second is the seat of oxidation and scorification, and the last only forms a passage-way and receptacle for the molten products. The most important region is the central one. It is situated some distance above the tuyeres, much higher up in the shaft than the slag-forming zone of the ordinary furnace, and fluctuates up and down in position in accordance with the variations in the pneumatic conditions within the shaft, its location being principally decided by the volume of blast blown in. To characterize it as that part of the shaft or column in which the vital functions of the pyrite furnace are in play, I have designated it the "focus." The three zones doubtless merge into each other, the lower preparatory zone and the upper reaches of the focus probably not being clearly defined. The lower margin of the focus, however, is probably sharply divided from the basal region. The furnace activity is all upwards, not downwards, the sinking of the fused products and the silica excepted.

The preparatory zone is probably not so high as in the ordinary furnace, and has the usual functions of driving out moisture and preheating the masses by means of the escaping gases. It has also a very special one, in the case of the pyrites, without which the further furnace action would not take place as it does. This consists in the sublimation of the volatile sulphur atom out of the pyrites, at the top of the zone, and the fusion and superheating, lower down, of the residuum, the composition of which, through the loss of about $\frac{3}{7}$ of the total sulphur, is practically that of pyrrhotite, Fe_7S_8.

In this uppermost zone no *chemical* action affecting the ores on charge takes place, and heat is only being absorbed, but it

appears that within this zone a chemical interchange is enacted, accompanied by the evolution of a small amount of heat, between the heated sulphur dioxide, coming up from below, and the coke fed in, as soon as the latter sinks to a horizon where it is hot enough. If the amount of coke is small, it probably does not survive this reaction, and no coke gets down to the focus. Should the amount of coke be large, then some may reach the focus, but it will necessarily interfere with the reactions there, and spoil the pyritic work. A small amount of coke on charge is, however, still a *sine qua non*, and seems required to support the work going on in the preparatory zone, even though its thermal effect is low, under the circumstances. As soon as the melting point of the pyrrhotite residuum is reached in this zone this substance melts, and we now have the phenomenon of an incandescent, honeycombed mass of silicious material in lumps, in the interstices of which aggregate the molten sulphide courses downward, towards the focus, in a heavy shower.

The atmosphere in the preparatory region consists only of the neutral gases forming the products of combustion from the focus, and the CO_2 out of the limestone and coke, and is heavily charged with sulphur vapour, derived from the disintegrating sulphide of iron and the reaction between coke and sulphur dioxide. By far the chief constituent is nitrogen; then follow sulphur dioxide and carbon dioxide. There is, or should be, an absence of oxygen, and consequently no roasting or other oxidizing action can take place in this zone. In the lower portion of the preparatory zone the limestone may be calcined, but this is more likely to be completed in the upper regions of the focus.

Physically, the preparatory zone represents a mere jumble of charge constituents undergoing certain changes. Yet the distribution of the constituents should be homogeneous and properly balanced, otherwise the furnace will not work well. Occasionally it may be necessary temporarily to disturb this regularity of mixture by placing more pyrites, or more silica, or more slag, here or there, for the purpose of curing irregularities in the descent of the masses, to relieve the beginning formation of

accretions, or to close blow-holes, &c., but even this treatment, though of a curative nature, cannot be persisted in too long.

It is a peculiarity of most pyrites to decrepitate with heat. Silicious ores, especially those containing a small amount of pyrites, do the same, and so does limestone, in addition to calcining. The Mount Lyell pyrites decrepitates well—that is to say, it neither remains inert in this connection nor does it fly into a fine powder. In both cases it would interfere with the proper furnace action, especially in the latter case, for this would mean packing the charge too tight. It contains no marcasite.

One of the first conditions for good work is to maintain the porosity of the column. As all our materials break up under the influence of heat, we are not under the necessity of crushing them down to conventional sizes before smelting. Everything is fed into the furnace in the largest-sized pieces which it is possible to handle or pass through bin-doors. The furnace virtually is its own crusher, and, while the iron sulphide soon melts and is put into a state of fine division by the current of gases, the silica keeps on breaking up in the descent, until it reaches the focus in small lumps of a suitable size.

The molten "pyrrhotite" is mercurial in character, being both heavy and very lively, and runs down freely in the interstitial spaces between, and in contact with, the silica. Its downward progress, however, is impeded by the upward tendency of the furnace gases. In fact, the latter form a pneumatic cushion for the sulphide, which carries it and prevents it from rushing ahead of the accompanying silica. The descent of the "pyrrhotite" cannot, as a matter of fact, be more rapid than that of the accompanying silica, notwithstanding its molten condition and greater weight, for otherwise the silica would be left behind out of each charge and the furnace activity proceed irregularly. Something akin to this seems to occur in cases where, the silica contents of the charge not being sufficiently "free," but the SiO_2 being combined with, or occurring in the presence of, bases with which it combines at the comparatively low temperature of the preparatory zone, a more or less fritted compound forms, which pastes the masses together and hangs up the furnace. Simul-

taneously the "pyrrhotite" liquates out and runs away as a low-grade matte. In this case there is no focus action possible, simply because, however proper the average chemical composition of the mixture may be, the requisite physical conditions are not supplied, and consequently pyrite smelting will not work. Most of the silica must absolutely be free and clean, and must reach the focus in fairly large, isolated pieces to ensure success.

On its downward course the "pyrrhotite" suffers a further desulphurization in the preparatory zone, in the hot current of neutral gases, under the influence of increasing temperature. It progressively loses more and more of its sulphur, until the particular combination of iron and sulphur, which finally reaches the focus, is even lower in sulphur contents than the monosulphide, FeS. Its composition naturally greatly varies, but for practical purposes it may be regarded as a solution of a little metallic iron in the monosulphide. Empirical formulæ, like 4FeS, Fe, and 3FeS, Fe, are commonly indicated as representative of this residual sulphide when the working of the furnace is investigated by calculation.

I do not, however, regard this high desulphurization as an advantage, but rather as a defect of our work—not on account of the possible liberation or precipitation of metallic iron in quantity, and the formation of "sows" (a phenomenon which has never been observed at Mount Lyell), but because the particular sulphur and iron compound which thus reaches the focus is the one that does all the work there, and, under the circumstances, the shortage of even the small amount of sulphur involved is a drawback. It has not yet been possible to regulate the composition of the sulphide that burns in the focus, and it is possible that, if we could currently burn a composition more like FeS there, the small amount of coke which it is still necessary to use to furbish up the general furnace heat might be curtailed, or altogether avoided. The point seems rather fine, and the endeavour savours of splitting hairs, but it is certainly inaccurate to take it for granted, as is usually done, that it is FeS that burns in the focus, when it is actually a less energetic compound.

From the foregoing it will be obvious that the sulphur in the original pyrites does not contribute very vitally to the heat developed in the pyrite furnace, notwithstanding its great quantity. The Mount Lyell pyrites now commonly has about 46 % of sulphur, and the North Lyell ore has 7.5 %. The ordinary furnace mixture of the two has about 30 % S. Yet, of the original total S in the FeS_2 treated (*i.e.*, not counting the sulphur protecting the copper in its passage through the furnace), only 36–38 % actually is burnt in the focus as a fuel. A bare 2 % of the total goes into the matte, and from 55–60 % of the original S in the FeS_2 distils out of the furnace as sulphur vapour, without combustion anywhere, except in the shaft above the column, where it comes into contact with the atmosphere. The temperature at the throat is high enough to ignite it. These figures indicate the loss of a very considerable portion of the total inherent fuel value of pyrites.

Unfortunately, there is no hope of realizing the calorific value of the sulphur thus escaping from the depths of the furnace, for the volatility of the vapour cannot be restrained, and the preferential affinity of the oxygen in the blast for the iron also discriminates against the combustion of the metalloid. Of the metal, iron, the furnace oxidizes quite regularly 95 % of the total contents in the ores. By contrast this represents a very valuable utilization of a constituent which is present in smaller quantity, and also has a lower calorific value, than the sulphur, for it saves the situation.

There is, similarly, no prospect of utilizing any portion of the loose (*i.e.*, volatile) atom of sulphur out of the pyrites, which at once sublimes off at the throat. The suppression of all these eliminations would only be possible if the tension of the furnace atmosphere could be sufficiently increased to counteract the vapour tension of the subliming sulphur. This, however, is beyond the reach of our pressures, and impracticable in our apparatus.

Were it feasible to burn the original FeS_2 in its entirety to the same degree as the Fe alone is now burnt, then the cokeless smelting would be accomplished. But it would have to be done

in an apparatus differing very considerably from our present blast-furnace (or converter), and it is not easy to forecast the constructional lines of a closed, high-pressure structure, which it would have to show. It would have to embody all the accessibility of the throat for feeding and barring which the blast-furnace presents, together with a sealed tunnel-head, and at the same time be as simple to handle as our blast-furnaces now are. On the other hand, the mere raising of the blast-pressure in the existing furnaces will not lead to the combustion of noticeably more of the sulphur. Unless the ore column is simultaneously increased, a very heavy blast will only cause a waste of wind, by escape from the top, and the formation of a low-grade matte, owing to the lack of preparation of the descending masses in consequence of the rising of the focus. Some of the escaping sulphur may be burnt in such a case, but this will be above the stockline. Such back pressure as can be arranged for in the present furnace construction by means of tight charging-doors, etc., is insufficient to keep down the sulphur to any extent, and inadequate to force it to remain fixed until it gets to the focus in greater proportion.

Something was hoped for in this general connection from the increase in height of the Mount Lyell furnaces. The column used to be 10–12 ft., and may now be carried up to 18 ft. No palpable effect has so far resulted from our efforts to cut down the coke and burn proportionally more of the sulphur. It must, however, be explained that just about the time the column was raised the percentage of Fe and S in the Mount Lyell pyrites began to decrease, while the Zn and Pb increased, and this change of composition has since become permanent. It has had an unavoidable deleterious effect on the heat developed by the charge alone, and, instead of being able to lower the coke, we have been obliged to raise it. We are satisfied, however, that, with the former lower column, we would be obliged to use still more coke, for the same blast volume.

The most important portion of the furnace activity manifests itself in the focus. This is situated several feet above the tuyere

level, at a point where the union of the two streams of blast, coming up from the row of tuyeres on each side of the furnace, may be conceived to meet, so that there is relatively the greatest concentration of compressed air there. It is, of course, impossible to tell how large the focus is, or how high it extends upward, but it is fair to assume that there is a comparatively sudden shading off from intensest activity in its central portion to lesser activity on its periphery, especially downward. The function of the focus is to fully oxidize the residual sulphide which trickles down to it, with the exception of the small amount which forms the matte and carries all the copper with it (assumed as Cu_2S). With this exception all the Fe is turned into FeO and all the S into SO_2. The former change, however, is not possible by itself, and takes place only in the presence of the incandescent SiO_2, with which the FeO simultaneously unites to form a silicate. It will, of course, be understood that I am reducing the matter to its simplest terms, and leaving out all refinements. What particular silicate of iron protoxide will form depends primarily upon the degree of heat in the focus, which, in itself, is the resultant of the heat developed by the combustion of the residual sulphide, plus the heat from slag formation, minus the heat absorbed in the dissociation of the residual sulphide and that required to break up and assimilate the other compounds present which are also incorporated in the slag. In all cases the action of the focus is absolutely self-governing, or autonomous. No slag forms but the one that corresponds to the focus temperature. The furnace chooses its own slag and sticks to it for the same set of conditions, and it is not possible to alter the composition of the slag by merely changing the relative amounts of the two ores. To effect a change it is necessary to alter either the amount of blast, or the limestone, or the coke. It is presumed, however, that we are already running with the proper volume of blast, that we desire to keep a suitable minimum of CaO in the slag, and that we are not anxious either to chill the furnace by applying too little coke nor to needlessly heat it up, and simultaneously to lower the grade of the matte, by applying too much coke. Pyrite smelting

does not foster pet theories as to slag compositions, and is deadly monotonous in this connection! The principal point, in this regard, about running the furnaces properly is to fully reconcile oneself to the restrictions which the focus action imposes, and to accept the inevitable, as far as slag composition is concerned. In the light of the other difficulties which beset the method, it must be accounted a very fortunate circumstance that, after all, a slag *does* form which is of the correct liquidity to run out of the furnace by itself without, at the same time, entangling too much of the metals which it is proposed to save!

The slag is necessarily highly ferruginous, and has the disadvantages which such a composition suffers from, as compared with a limey slag. Comparisons with the latter are, however, quite beside the mark. It would be most uneconomical to force CaO into our slags, for high lime lowers the oxidation and scorification and increases the coke, with the attendant evil of low concentration, etc. Too much limestone also slows up the furnace. The utilization of the iron in the pyrites as the principal flux is the only rational procedure under our circumstances, whatever may be correct in semi-pyritic smelting. The iron flux employed is derived from the crude pyrites direct, without roasting, and acts as its own fuel, to the exclusion of nearly all the coke, and, in addition, it is an ore which introduces gold, silver, and copper values, whereas limestone has no similar virtues.

Even in the focus the general porosity or vesicular structure of the column must be maintained, so that the action may be unhampered. The solid portions of the charge here practically include only the free silica, and of this the original lumps are much reduced in size by decrepitation by the time they get to the focus. Yet there is no packing of the pieces, for else they would not present the necessary surface to the combined attack of the air and the residual sulphide. A certain amount of mobility is doubtless fostered by the circumstances that the combustion of the residual sulphide is accompanied by the explosive evolution of a large volume of highly-heated sulphur dioxide, and that the

union of the FeO and the SiO_2 results in the production of a fused substance, the slag, which occupies much less space than did the original materials which formed it. It is fair to assume that, if the solid materials are only loose enough not to pack tight in the preparatory zone, they will not pack in the focus. There is probably never much danger of a furnace packing in the focus. The activity there, being entirely self-governing, goes as far as it is compelled to do by the chemical laws obtaining and then stops, if there is no further food for activity brought to it from above. All the sulphide that comes to it molten, but which it cannot oxidize, simply runs away. There is no half-way mixture of things in the focus even of a frozen furnace—no mushy or gummy mass—no half-cooked slag or half-molten matte. When the relation of things is no longer proper for the focus action the latter at once ceases, and the focus abolishes itself, leaving only the silica skeleton behind in its place.

One of the most important points in connection with the running of the furnace is to see that it is properly open above and below the focus, and the steps to accomplish this constitute the special task of the furnace crew. A prime necessity, as already remarked, is to use all materials in as coarse a condition as possible, and it is necessary to counteract the fineness of portion of the charged materials by the coarseness of the balance, to the extent of which the supplies allow. For this purpose much (and instantaneous) manipulation is required, and the attention given is never relaxed. The proportion of fines in the Mount Lyell pyrites is, on the whole, small, but there are runs of it now and then which are awkward. The North Mount Lyell ore formerly contained fine schisty stuff which was deadly to put through, but, on the whole, this mine has been getting more densely quartzitic with depth. Large lumps of dense schist occasionally find their way down from both mines, and are spawled on the floor before charging if they contain ore, and are rejected if they do not. Though limestone tends to increase the openness of the charge, it is not applied in extra quantities for this purpose. A certain fairly large amount of slag used formerly always to be put on charge, as so-called

"ballast-slag," to facilitate the furnace action and contribute to openness. As it takes coke to smelt it, however, this practice has been given up, and the slag on charge much reduced. The use of slag charges—*i.e.*, doses of slag only, with the requisite coke to smelt it—is avoided, and may only be used in the case of extremity. They are costly, and no credit to those in charge, as long as other means to keep the furnace open can be applied.

The system of bedding the ores in large quantities prior to feeding does not seem to have any application to our case. There are so many important variable factors controlling the work, only a few of which the bedding would turn into constants, that it is doubtful if anything would be gained. Notwithstanding that coarse and fine would be properly mixed, it is still likely that the cry for clean, coarse ore would occasionally arise on the charge-floor, in anxious moments, and so bedding appears superfluous. From the point of view of an even distribution of the copper values, nothing is to be gained from bedding, for the fluctuations in concentration ratio and oxidation factor are too frequent and uncertain for a perfectly even ore-assay to have any special advantage.

Particular efforts are uninterruptedly directed towards the maintenance of a proper silica column in the furnace shaft, for this is the soul, or rather the backbone, of the operation. It is difficult to keep in touch with this column, and, as the nearest measure towards it, it becomes imperative to particularly watch the throat phenomena, especially the freedom with which the charge column sinks. The throat of the furnace is a much more prolific source of troubles than the tap-hole, and, provided other things are fairly right, the maintenance of a freely moving throat is a guarantee that the furnace will smelt well. As far as possible, the formation of accretions should be avoided, and this is really one of the most difficult tasks one is confronted with. By far the majority of the troubles encountered with pyrite smelting in its earliest days, during trials in small furnaces, were due to the hanging of the charge in the throat, and the formation of accretions. It would take too much space to go into details, but

it frequently happened that the slag would suddenly stop running, and upon investigation it would be found that the furnace had smelted itself empty, leaving a hollow cavity in the shaft, below a bridged-over throat carrying the last charges put in a long time before. No amount of blowing would resuscitate this condition of things—in fact, further blast would only turn the furnace into a forced draught roasting kiln, until even this action ceased. At other times the scaffolding was simply due to carrying too high a column for the weak blast available, so that the preparatory zone was too high, and the volatile atom of sulphur had a chance to condense and glue the masses together in the throat.

Ordinarily with us, however, throat accretions are caused by improper feeding. The small amounts of zincblende and galena in the pyrites cannot be accused of contributing much to the trouble. Proper feeding, on the other hand, can be made to resuscitate a furnace which has been closing up. The accretions are, of course, barred off mechanically from time to time, as in other work, if necessary. No attempt is made to smelt them out with coke and slag.

The most important operation in connection with our work is the feeding of the furnaces. This is done with special care, though it still remains rather rough work, in a sense. As the throat is fiery it is not approachable, and the fine work done in the olden days, when throats were cold and the blast low, cannot be carried out now.

It is being conceded that the manner in which the modern copper blast-furnace is fed, by simply sliding the charge into it *en masse* over an inclined plane on the long wall, is too crude to do justice to the requirements of good smelting. It certainly shows itself to be a mistake in pyrite smelting, and the mechanical features, which make it attractive to the metallurgist who is only bent on saving the small labour attaching to the actual feeding, are blinding him to the troubles and costs which the bad feeding entails, and which are not so obvious. It is quite possible to deliver the materials to the furnace by the same means (electric train) in both cases, but they should not be dumped in helter-

skelter. Whatever may be the case with ordinary blast furnaces, it is a fact that pyrite furnaces, on which this careless method of charging is used, are not doing good or really economical work, as is evidenced by their low-grade mattes and short campaigns. It is impossible, in the first place, under the conditions of wholesale delivery, as mostly arranged nowadays, to do close work, such as running each furnace individually, and the result is that the nest of furnaces is run as one unit. This obliges one to average the composition of the charges practically down to the mean requirements of the nest, and to forfeit the advantage of the superior work which can be done by the furnaces which are in better trim. Those which are not in good trim cannot, of course, be forced to do good work, and a mediocre standard, embracing all the furnaces in the nest, is the only recourse. Dump feeding, whether from the side or from the top, treats the operation as if it had to be got over as quickly as possible, and as if the amount of care required were negligible. Our experience proves that this is not so. For a lengthy period we have, moreover, ourselves used an automatically discharging dump-car, which dropped a whole charge at once on to the column from above, but the furnace did not smelt well, and the method was discarded, because too little control could be exercised over the feeding. The furnace even had a low tonnage capacity, in consequence of the tightening of the column by the falling charges.

The feeding at Mount Lyell is done on each long side of the furnace, off a horizontal charging plate, which reaches into the shaft with a slight overhang. On to this plate the charge constituents are tipped, out of end-discharging hand-carts, in a narrow pile, as long as the opening and parallel to the furnace, so that, when pushed over the edge of the plate, the materials drop so as to cover one-half of the top of the column on each side. The materials drop gently over the edge of plate, and do not shoot across to the other wall and rebound. A special device is used for pushing the substances in, which consists of a line of hinged steel plates, inclined forward and parallel to the furnace, which is fixed to a frame embracing both sides of the furnace, and is

actuated by means of an hydraulic cylinder. The movement is to and fro, the two sides of the furnace being fed alternately. The pusher plates ride freely, on the return stroke, over any matter left on the charging plates, but the latter are swept quite clean on the forward movement. The charge constituents are not delivered mixed together, but singly, one after the other, and are pushed in separately. No special merit is claimed for this device, but it certainly has simplified and perfected the feeding. It is possible to do with it all that can be done by hand feeding, and, when particular places on top of column have to be specially humoured, the feeder still can do so by hand, if he likes, with the long-handled shovel, using the blade inverted. The distinction between coarse and fines can be maintained as with hand feeding, the respective stuff being directed to be dumped by the wheeler in front of the right spot, or the separation can be done by the feeder on the plate, with a few strokes of the shovel in the long heap, before pushing. The placing of the materials on to the charging plates is under the immediate direction of the feeder, and he is held responsible for the condition of the throat. If the throat is properly nursed the rest of the ore column in action will take care of itself.

A good deal depends upon the placing of the coarse or fines at the proper place on top of charge, and in connection therewith the proper overhang of the charging plate is of some moment. Notwithstanding that all materials fall almost vertically off the edge, the fines fall closer to the wall than the lumps, and the exact place where the latter assemble depends upon the vertical distance from charging edge to top of column. The furnace can be fed with a gullet down the centre line, or with a hillock along that line, or with more than one gullet or furrow. The overhang distance has an effect on these points, and a device has been worked out permitting of instantaneously changing the amount of the overhang, but it is not really being used. It would lead too far to go into further details. Ordinarily speaking, for normal running the practice is to distribute the materials evenly over the full top of the column. The latter is not visible from the

charge openings, as these are made only high enough to allow the charge to be shoved in (14 in. by 18 ft. 6 in.), so that the indraught of air and the back-leak of fumes may both be minimized. There are no doors on the charge openings. The feeder observes the state of affairs at the throat through a lidded opening in the swinging doors at each end of the superstructure. The barring of the furnace throat is done through these doors only.

The gases must be seen issuing evenly from all over the top of column, with only slightly more coming up along the walls. The pieces of charge, of course, remain black on top of column for some time after charging, and those of the underlying layer show a dull red heat. There is no appearance of high incandescence. A perfectly black top would soon lead to the formation of crusts. The general temperature right over the top should not be much higher than suffices to ignite the subliming sulphur, and the top must be kept down to bring this about, but not much lower than this, if possible. There should be no blow-holes. Dead, glowing red places are indicative of the formation of accretions, or wall crusts, and are not permitted to grow too much. They would eventually close the furnace throat, and, in any case, they upset the distribution of the blast. They are attended to at once by feeding some of the pyrites on charge on to them, and keeping the silica off. If accretions have formed all along the walls, then most of the feeding is done along the centre line of throat. When the throat is working properly there is a constant crackling noise from the decrepitating of the pyrites, etc.

There is a fixed order in which the constituents of the charge are fed in, which is never varied. The coke is first put in, then all the pyrites, then the silicious ore, followed by the limestone, and finally the slag. The pugged flue-dust and the barren silica are put in at intervals, as dictated by the supply, or as required by the furnace conditions. They are not made regular constituents of the charge. The coke, being small in amount, scarcely falls so that it covers the whole of the respective half-surface of the top, but comes nearer the walls. The other materials, as

remarked, are each equally spread all over the top when things are normal. A constant source of derangement are blow-holes, which interfere by leading the blast preponderatingly to themselves, and robbing it from the rest of the furnace, thus disturbing the uniformity of the focus action. They are cured by filling them up with the "dry" portions of the charge—*i.e.*, North Lyell ore, slag, silica, and limestone, as a general rule.

An important essential is steady running. Frequent, and especially long-continued, stoppages are fatal. A prime condition for good work, next to coarse ore and free silica, is a reliable, constant, and sufficient blast supply. The speed of the furnace and the grade of the matte depend upon the amount of air blown in. The item of pressure is merely incidental to the volume of blast and the resistance in the furnace, and the latter is governed by the height of column in an open furnace. As a general rule, the actual practice is the same as in bessemerizing in converters—namely, to blow as hard as one can, within limits. These limits are the oxidation of a sufficient proportion (95 %) of the residual sulphide, plus the coke, and to have, if possible, all the air absorbed in the column and no free oxygen escaping at the top. The inner mechanical condition of the furnace decides the quantity of air consumed by it. If packed, it will take less air, but show a higher pressure, and if free and open it takes the air normally. If blow-holes exist they lead to a great waste of air. We now supply ordinarily 17,000–19,000 cub. ft. of free air per min. to a furnace, and the average pressure, at present, since the hydro-electric scheme has been installed, is maintained at about 72 oz. per sq. in. in the blast-main, and could be raised to 8 lb. The centrifugal blowers supplying the blast deliver it in a perfectly constant stream, and this has been found most beneficial. Even the best regulated steam-driven power-plant, including the high-class steam-turbine-driven turbo-blowers supplanted by the new ones, is subject to irregularities of running, emanating from the frailties of the boiler-plant, which require skilful manipulation of the team of engines and furnaces to achieve the most continuous running. The new power makes

this an easy matter. The practice has always been to run the blowers with constant speed, so as to supply a constant volume, as required by the number of furnaces running, and to let the pressure take care of itself.

The three important points about the volume of blast to be used are, first, to achieve a sufficiently intense focus action to yield a hot and liquid slag, which will flow freely from the furnace and through the chain of forehearths, and will allow the matte to settle out of it cleanly; second, to get a high rate of concentration, yielding a matte of a copper tenor suitable for cheap converting; and third, to accomplish as complete an absorption of the oxygen of the blast as is possible, preferably 100 % "oxygen efficiency" of the furnace. The temperature of our slags (and matte) at the sump spout is from 1100° to 1250° C., and the copper tenor of the matte is from 40 to 50 %, while the oxygen efficiency of the furnaces is such that, in normal running, the oxgyen in the gases issuing from the top of column is practically *nil*.

The furnace gases have often been analyzed, and the most recent determinations have been made with a type of apparatus which fully meets the imputation that our analyses may have been wrong, because the oxygen, which we could not find in the tests, may have combined with the iron of the pipe by means of which the gases were drawn off the furnace below the stockline. The percentage of SO_2 varies considerably over the area of the throat —from 4 % to 12 % by volume—and normally seems to be about 6–8 %—but it does not appear to vary with the depth at which the gas is extracted—that is to say, there is no noticeable increase or decrease of the SO_2 as the focus is approached. As the chemical reactions are complete in the focus, this fact is comprehensible. The gases seldom show any CO. The CO_2 is about 6.0 %, and the nitrogen is very high, being practically the difference between the sum of $SO_2 + CO_2$ and 100 %. The greatest depth so far tested below top of column is 7 ft.

The gases naturally escape with a considerable amount of heat, but in normal running this is not really excessive, though the throat may appear very fiery. As remarked, the throat is so

managed as to cause the combustion of the subliming sulphur, for this would otherwise be a nuisance in the way already indicated. The fact that the combustion of this sulphur above the stockline creates a high temperature there has to be mentally dissociated from the lower temperature of the furnace gases proper. It is not a part of the furnace reactions.

The height of the furnace column now used is up to 18 ft., as already mentioned. The height is regulated by no consideration so much as by the avoidance of a sticky throat on the one hand and the keeping down of a general overfire on the other. In other words, practically, the height of column is a matter of contest between the sticking quality of the charge and the oxygen efficiency of the furnace. The turbo-blowers used are in great part responsible for the increase in column, but it cannot be said that we have even now reached finality in this matter, though, for constructional and related reasons, we are satisfied with the 18 ft. above tuyeres as a maximum. The variation in height of column amounts to about 4 ft.

The furnace tuyeres are oblong, and measure 15 in. by $3\frac{1}{2}$ in. (as long as the cast-iron jackets will allow), and approach a continuous slit around the furnace. They are pointed slightly downward, principally for the reason that they thus do not so readily fill, and discharge matte into the tuyere boxes, when there happen to be local runs of this substance down the jacket wall. The tuyeres are always dark, and the desire to inspect them visually has long ago been relinquished, the "peep-holes" in the tuyere boxes being closed with wooden plugs instead of mica windows. A light in the tuyeres is not welcome, for it simply shows the proximity of pockets of matte, the proper place for which material, at the level of the tuyeres, is the centre line of the furnace. These oblong tuyeres are now scarcely ever touched with a rod, for punching them serves no purpose. The hearth of the furnace is, of course, filled with a mass of inert material, and this chills into a shelf-like prominence at the tuyere level, on either side along the jackets, but is doubtless highly heated in the centre line, where the matte and slag come down. It is very

doubtful, however, if, in this region, any further heat is communicated to the products, for the oxidation is likely to be negligible there compared with the focus. Anyway, the atmosphere prevailing in the tuyere region has been found to be only common air, without a trace of SO_2. It is, furthermore, possible to drive a bar clean through the furnace, from tuyere to tuyere, and withdraw it, without its being corroded or getting greatly heated up!

As there are no explosive gases formed there is no necessity for safety valves on the individual downcomers from the bustle-pipe to the tuyere boxes. This wind-connection I have always made as short as possible, to avoid unnecessary travel for the wind. Although the jackets are sectional cast-iron ones, this design does not interfere with their replacement. The furnace cannot be stopped merely to renew a jacket or two, and they are put into position when the furnace is out of commission, from the inside.

The furnace in action forms its own smelting profile, which is roughly wineglass-shaped. Its lines fluctuate with the pneumatic and other conditions, up and down, and the spaces in the shaft between the wineglass contour and the jackets fill up with inert matter, chiefly chilled slag containing pieces of silica. These crusts form the furnace shaft proper, and, as they are relatively thick everywhere except at the focus, where the bowl of the glass is, the jacket-cooling water does not require much attention, and the jackets are run very cold; in fact, in order to minimize the water consumption, the jackets are interconnected vertically, the water traversing more than one row of jackets. The only jackets which require particular attention as regards cooling are those at the height of the focus. This row cracks the oftenest, and is best provided with cooling water. The water connections are complex, and there are many valves to attend to, though the fact that the jackets are run very cold eases the work. The water supply to jackets is in the hands of special handy men, holding the premier position on the lower floor as "furnace men," with a knowledge of tapping-floor work, who attend to the water on more than one furnace, and assist in front of the furnace in

emergencies. There is, of course, an unrestricted supply of water available. The valves are made of a non-corrosive alloy, and the supply pipe-connections are extensively used in lead, to facilitate bending. Leaking jackets are not replaced until there is a large number, and the blowing-out of the furnace for repairs is justified.

The furnace campaigns have long ago outgrown their initial brevity, and there is now seldom any other than a purely mechanical or similar adventitious reason why furnaces ever stop. Campaigns of six months are achievable, and a furnace has remained in blast for more than a year even. Constant watchfulness is a *sine qua non*, and every device is resorted to to keep a furnace in good running order. On the other hand, no waste of money is incurred in keeping it running when the disorder can only be cured by means of coke and slag charges. It is much cheaper to blow out at once, clean the furnace out, and start up again, than to try to smelt a furnace into order again. If unforeseen stoppages, such as a breakdown of the blast, occur, then the furnace is tapped dry and banked. Pyrite smelting is inconveniently sensitive to interruptions, for reasons that will be patent without special mention, but unexpected stoppages of a couple of hours are nowadays no longer a calamity—an advantage derived from the size of the furnaces.

The character of the slag is that typical of its particular composition, pyrite smelting introducing no difference in this respect from coke-made slag, except that the pyrite slag is remarkably uniform in appearance and analysis, more so than the other. The correct slag is hot enough to run through the main forehearth and the three or four small auxiliary forehearths, thus describing a path of about 40 ft., and still show the well-known characteristics of a hot, highly ferruginous slag, with at present 36–38 % SiO_2. The fall in temperature between sump, or furnace-discharge, and the last slag spout is only about 80–90° C., and, collected in the pot at the end of this run, the slag is still brilliantly bright, and smokeless, and rises perfectly free all round the rim, with a shining line, and with a leathery crust on top, showing the

typical iron markings. The copper contents of the slag bear the usual relation to the grade of the matte, the assay running from 0.25 to 0.5 % Cu. The sulphur contents are very low, considering the Fe and Zn, being 0.3–0.6 %. The slag going through the small forehearths sheds a little matte in each one, which is tapped several times a day.

The waste-slag stream falls direct into an ordinary inclined launder, sunk below the tapping-floor, and is granulated and washed away by the accumulated jacket water of the respective furnace, with additions, and there is no special device for the granulating. The stream is simply struck by a ½-in. jet of water coming from an elevation of about 120 ft. The slag from all the furnaces runs to a common pit in the yard, out of which it is now being elevated to a height of 90 ft. above the tapping-floor level by means of a substantial bucket elevator, in a much-dewatered condition. From this elevation it is again sluiced away by means of water on to the top of the slag dump. Until recently the disposal of the granulated slag has been carried out by means of centrifugal pumps, which, however, could only lift the slag, plus all the water, and so required considerable power.

The tapping-floor operations do not differ from those in vogue in copper matting plants generally. The slag, being run off continuously, requires no special attention except keeping the spouts clear. The matte is tapped out of the main forehearth at intervals, as usual. The concentration ratio being high, there is not really much matte made, but there are occasional rushes of it, as the outflow is not altogether regular.

The cleaning out of a furnace, after blowing out, is a simple and easy matter. The matte having been tapped out of both ends of the furnace, the latter is "dry," or bare of matte which might cause much labour for removal of chilled masses in the crucible, and the balance of the material in shaft is so soft as to be readily broken and shovelled out. It is merely a honeycombed mass of slag with pieces of silica embedded in it, except up above, where there is a loose aggregate of the masses of the preparatory zone, surmounted by the remnants of the last charges put in. In

the region of the focus there are no signs of chilled products of fusion, except the slag, mixed with the pieces of unfused silica in the usual way. The side crusts, in the hearth and above. are of the same nature, and all is easily shifted. The throat accretions are tougher, as they contain sulphides, and are largely only half-formed products.

The inner working profile of the furnaces is not so plainly visible after blowing out at the present time as it used to be in the former smaller furnaces.

The time required for blowing out and starting up again is very short, unless the mechanical repairs required are extensive, such as replacement of jackets, repairs to water connections, &c. The record from beginning of blowing out to blowing in again, inclusive of the replacing of some broken jackets, is 16 hours, and the ordinary time is about 36 hours, and the corresponding cost is very low. For this reason no hesitation is felt in blowing out a furnace at once and starting up fresh, if anything goes seriously wrong with the smelting, for more time and money (for coke and slag, &c.) would be expended in bringing the furnace round metallurgically than by this procedure. This applies also in cases of bad blowing-in.

Notwithstanding the sensitiveness of the process, there are no "freeze-ups." There is also no formation of "sows." The latter might be due to the precipitation of masses of magnetic oxide out of over-oxidized slags, or to the shedding of metallic iron out of the residual sulphide in quantity. Neither of these phenomena has been observed in our furnaces, except perhaps the first-mentioned, locally in the hearth, to an insignificant extent. Occasionally there have been what we call "silica sows," however. These occur when the charge contains an overdose of silicious ore, over and above that which the focus can deal with, owing to relative shortage of oxidizable iron in the charge. The slag and matte normal for the conditions form and run out, but the overload of silica gradually tells if the defect is not remedied by taking off some of the silicious ore, until the surplus of undigested silica chokes the furnace, and the heat supplied by combustion and

scorification becomes inadequate for the heating-up of the accumulation, and the furnace slows up and gets cold.

The matte and slag are frequently sampled, for it is necessary to keep in touch with the copper contained in them, especially in the matte. In fact, the furnaces are entirely run on the showing of the matte assays, and thus the ordinary position is reversed. Instead of calćulating, or making up a charge of a composition suitable for producing a matte of a certain tenor, which the furnace is expected to supply, the ores are put in in proportions which are known, empirically, to yield the grade of matte desired, and the matte assay is employed to institute such varying changes in the composition of the charge as may momentarily be necessary to keep the matte tenor approximately at the mark fixed upon. In other words, the furnaces are run "backwards," in the sense that the grade of matte falling governs and regulates the composition of the charges put in.

In the case of the matte, rod samples are taken regularly, at intervals of three hours, and as much more frequently as may be necessary. The samples are at once analyzed by the shift-bosses, who are taught to do the Parkes cyanide test for copper, on simplified lines, with the use of proper chemical apparatus, however. Their method consists in weighing out 200 mg. of the ground matte, boiling in a beaker glass, with 6 cc. of nitric acid, down to the elimination of fumes, taking up with 50 cc. of water, adding 12 cc. of ammonia, decolourizing with a few cc. of a standardized potassium cyanide solution prepared in the laboratory, filtering, and finishing the titration in the filtrate, the usual burette being used. The results are sufficiently correct to control the furnace work by. For the metallurgical records, for office purposes, another matte sample is taken, by means of the rod, out of every third pot of matte tapped, and the weekly accumulation of such samples is assayed as one lot for all the furnaces.

Of the slag, regular daily samples are hourly taken from each furnace by means of small spoon samples off the slag stream as it drops into the granulating drain. A daily average analysis from each furnace is all that is required for current information.

The personnel of the furnace crew consists, per shift of eight hours, on the feed-floor, of one feeder and five wheelers to each furnace, and a feed-floor boss over them all. The maximum load handled by a wheeler is half a ton, which is the fixed weight of pyrites charged at a time. In order to balance the amount of labour evenly among them, these men employ a system of rotation of their own. Light, well-balanced, end-tipping carts, running on two large wheels, are used. Much thought has been given to the subject of replacing the wheelers by some system of mechanical conveyance, but the difficulties are insuperable, in view of the fact that it is necessary to have the utmost elasticity in the delivery of the materials. It is necessary to be able to supply any ore, at a moment's notice, from any bin to any furnace, and to be assured against absence of intelligence and breakdowns on the part of the mechanism into the bargain. On the tapping floor there are, per shift, one tapper and one helper per furnace, who attend to matte and slag, and one head furnaceman for all the furnaces, with an assistant. The further supervision consists of a shift boss on each shift, for both floors, and a general foreman for the plant.

More particularly, the question of charge composition turns about the following considerations:—

It will readily be understood that a natural consequence of the treatment of the same two ores for so many years is that an extensive fund of experience has accumulated in connection with charge compositions, and that a methodical system of "rotation" of charges is in use. The two ores are chemically complementary —the Mount Lyell pyrites being basic and the North Mount Lyell ore acid. They break large in the mines, barring some fines, and there is no necessity for crushing them. This operation is restricted to the aliquot portion which is held out for sampling purposes and passed through the sample-works. The character and metal values of the ores being remarkably uniform, the quantity thus dealt with is reduced to a minimum, and consists, in the case of the pyrites, of every 25th bucket (11 cwt.) coming over the aerial ropeway, which transports this ore, and, in the

case of the silicious ore, of every 12th truck ($1\frac{1}{2}$ tons) coming over the surface haulage which delivers it.

Besides being coarse, both ores have the very special feature that one is straight pyrites and the other is practically straight silica. This is an immense advantage for pyrite smelting, for it contributes both to the porosity of the column and to the facility of contact of the three factors coming into play—the sulphide, the free silica, and the blast. The chemical composition of the material smelted is not the only desideratum. A single ore having exactly the average composition of our two ores—that is, one consisting of just the right mineralogical mixture of pyrites and silica—would be very much more difficult to treat, even if used in coarse lumps. In addition, our ores have no constituents which might prejudice the chemical reactions. The zinc and lead are present in very small amounts, and heavy spar is subordinate. Alumina is not high enough to do any harm. There is also very little arsenic. Neither are there any injurious combinations of silica with earthy bases, &c., which might disturb desirable conditions in throat or focus. We occasionally treat small quantities of refractory ores, but their influence is negligible. Our principal bugbear is fines. There are also certain varieties of North Mount Lyell ore, of a schisty nature, which give trouble. They break in long, fish-like pieces, which lie flat and dead in the furnace, and consist of a fissile, talcose material, carrying more than the usual percentage of alumina.

The standard charge of the past was one on which two parts of Mount Lyell pyrites were treated, together with one part of North Mount Lyell ore, the actual quantities on charge being respectively one ton and half a ton for each side of the furnace. This weight of pyrites has been retained, but the silicious ore has been increased. Then, as now, a conventional amount of 250 lb. of limestone was added, and a fair amount of ballast slag, with coke to suit. Frequently, however, much less North Mount Lyell ore was used. The matter really depended upon the output of the two mines, and the intended copper output. As long as the pyrites was good in copper it was smelted in quantity,

and, if necessary, barren silica was employed to help out the silicious ore. As soon as the copper value of the pyrites decreased, and particularly since the reserves in the North Mount Lyell mine have increased, the smelting policy had to change, and it is now our desire to treat a minimum of pyrites together with a maximum of North Lyell ore. The process, however, imposes certain restrictions, which do not allow of going too far in this reversal, and the question of costs and profits also enters. It is not possible, for instance, to smelt equal quantities of Mount Lyell and North Mount Lyell except at greater cost and less profit than at present, and our treatment would have to change altogether into the employment of heavy proportions of limestone and coke.

The principal objective in the furnace work is the achieving of a good grade of converter matte (45–50 % Cu), and to do this the charges are constantly being altered, occasionally several times a day. The various furnaces in blast usually work on different charges, though treating the same parcels of ore. Local experience has supplied a lengthy schedule of charge compositions, and the particular "change" which is required on the evidence of the matte assay, and the ore and furnace conditions generally, is reliably fixed upon. The "change" is "down" in a couple of hours, and the matte is again assayed. If not correct, further changes are made in the charge. The situation requires constant watching, because, owing to the incessant variations in the physical and pneumatic conditions in the furnace, the tenor of the matte is also continually subject to deviations from the normal. Wide limits are fortunately set for the extremes of tenor —*i.e.*, from 40 to 50 % in copper—but even then care is taken not to allow the lower limit to reign too long. The slag composition is stable in comparison with the matte tenor, and is not affected by the changes in charge.

In connection with the latter, the most potent factor is the free—*i.e.*, uncombined—silica on charge. It is this element which governs the grade of the matte. The North Mount Lyell ore has a good excess of free silica, and is excellent to use. With the

deepening of the mine the proportion of clean silica has rather increased. The upper levels yielded a more schisty ore, and the chemical behaviour of this, in the furnace, was not so incisive, so to say, as that of the ore of to-day, the free SiO_2 having been less. The practice in regulating the grade of the matte is simply to vary the weight of North Mount Lyell ore on the charge. If the grade is not high enough, the weight of North Mount Lyell ore is increased, and if the grade is too high this weight is diminished. Experience is sufficient to tell how many pounds of the ore it is necessary to put on or take off. The same effect cannot be obtained by varying the quantity of pyrites and leaving that of the silicious material constant. Experience teaches that, in this case, the action is not so sharp and decisive, and the effect more uncertain. Besides, the variations in the pyrites weights would have to be much greater than is required with the silicious ore weights.

With respect to the instance where the North Mount Lyell weight has to be increased for the purpose of raising the grade of the matte, it may be argued that this improvement is effected because the additional silicious ore (which contains about 6 % Cu) simply introduces more copper into the furnace. This copper, however, is naturally combined with a sulphide of iron, and, consequently, the amount of matte would simultaneously be increased by its introduction, and the tenor of the total matte not essentially affected. Moreover, the extra silica coming into the furnace would, on this theory, simply have to join the slag formed under the former conditions and raise its SiO_2. This, however, is not the manner in which the pyrite furnace works. It is impossible for it to assimilate additional SiO_2 in this fashion. One of the earliest lessons to learn was that the furnace was intent on making its own slag composition and stubbornly insisted on maintaining it. The addition of more North Mount Lyell ore does not raise the SiO_2 percentage in the slag. On the contrary, the SiO_2 and the FeO in the slag both remain the same as before in percentage, notwithstanding the addition of the silicious ore to the charge. What happens is, that *more slag* of the old com-

position is made, and that more iron out of the pyrites is oxidized and scorified by the extra silica so as to bring this about. The additional SiO_2 simply induces a greater proportion of the fixed quantity of Fe present in the pyrites to enter into combination. The result is that less of the Fe is available for going into the matte, less matte is made, and the smaller matte-fall is, by a mathematical necessity, higher in grade. The copper in the additional North Mount Lyell ore joins the copper present before, and helps in raising the tenor of the matte, but the effect of the added SiO_2 is much more trenchant. The utilization of the oxygen in the blast is greater in the new charge—*i.e.*, the addition of the extra SiO_2 raises the oxidation factor of the furnace. It is not, however, the practice to blow more air into the furnace for this purpose; the amount of blast delivered from blowers is not varied for reasons of this kind, but ordinarily kept at a constant maximum point during normal running. The time required for a change in grade of matte to show itself is generally identical with that required for the "change to come down," as the expression is, though sometimes it is longer.

The variations in the composition of the charges run between the following extremes, which include the blowing-in charge :—

Mount Lyell pyrites, always constant	2240	lb.
North Mount Lyell ore	1000–1700	,,
Limestone, constant	250	,,
Slag (mostly from converters) ..	50–550	,,
Coke	160–250	,,

The endeavour always is to use as much North Mount Lyell ore, and as little slag and coke, as possible, and corrections of the charge are always going on to accomplish this without injuring the grade of the matte, or some other point about the smelting.

The daily tonnage furnaced is 1000–1200 tons (2240 lb.) of both ores, with an average of about $2\frac{1}{2}$ % Cu, into converter matte of a grade of from 40 to 50 % Cu, average about 45 %, in the one smelting. This high ratio of concentration is achieved as a rule and currently, though not infallibly. Mattes as low as 25 to 35 % Cu sometimes fall, and even as low as 15 % Cu; but the last is

exceptional, and none of these grades is allowed to form for any longer time than can be helped.

The limestone on charge is kept constant, for the simple reason that to vary it would be a superfluous refinement. Though the quantity on charge is slight, it cannot be left off without injury. Local experience proves that slags devoid of CaO carry too much copper, and that about 4 % CaO is amply sufficient to prevent this. The lime interferes with the avidity of the free silica for the iron, and higher lime slags, besides being costly, lower the oxidation factor, produce larger falls of lower grade matte, require more coke, and are not necessarily a guarantee against the retention of copper in the slag, because the matter of specific gravity is not the only point that enters in this connection. There seems to be a CaO content at which the copper again enters our slags. Very limey slags, of course, are beyond the scope of pyrite smelting. The limestone used is only of fair quality, and very variable in the percentage of CaO. The deposit quarried is situated three-quarters of a mile from the furnaces, and, though not extensive, has served the company for the last 19 years.

The amount of slag on charge is governed by the supply, and is also regulated according to the furnace needs. It consists mostly of the slag made by the converters and the shells from the settling pots in front of the last small forehearth. It was formerly employed in greater quantity than now, as "ballast-slag," and had to be specially saved from one furnace. The quantity at present available averages about 150 lb. per charge.

The amount of coke on charge requires the most control and regulation next to the free silica. Every opportunity is availed of to cut it down to the lowest limit—that is to say, when a furnace is doing well it is seen whether it will not run just as well with less coke. The number of changes made is not so great as in the case of the silicious ore, but there may be several a day. The coke is never changed for the purpose of influencing the grade of the matte, and, in one respect, this would be wasteful. The more coke the less oxidation of iron, and the lower the grade of the matte; and obviously, in such a case, the coke should, if possible, be reduced. But if, as sometimes happens, a furnace

makes too rich a matte, it would not be prudent to reduce the grade by means of more coke. The function of the coke also never is to loosen up the charge. It is simply to supply heat and temperature, and this purpose has to be taken into consideration, compounded with all the other points that enter simultaneously, when selecting the proper charge for the case in hand. The coke is produced from a somewhat dry coal at the company's own coke works at Port Kembla, N.S.W. Owing to shipping conditions, it has to be carried in bags, but these are at once emptied on arrival into a storage bin, close to the furnace-plant. It gathers considerable moisture *en route*—about 10 %. It is purposely made very hard and dense, yet, though shipped in large lumps, often arrives rather fine. The ash is about 16 %. The average coke consumption is now as follows :—

	Gross.	Dry.	On carbon contents.
Per cent. coke on ore	5.35	4.93	4.15
,, ,, ,, burden ..	4.55	4.17	3.51

As frequently remarked, the coke consumption is now very much higher than it used to be in the past. It is no longer possible to get on with as little as $1\frac{1}{2}$ % coke on the charge. During the hot-blast regime the consumption was even as low as 0.1 % coke on charge. The explanation of the present higher consumption is as follows :—We are no longer in the position to smelt as much Mount Lyell pyrites as is necessary to attain these low coke percentages, and, in addition, the pyrites has changed in chemical composition to an extent which, though slight, is sufficient to cause the difference in coke. The alteration in conditions can be readily followed, and the necessity for more coke proved, by a metallurgical calculation, but it is not necessary to go into same here. The falling-off in the iron contents in the pyrites is from 45 and 42 % Fe down to as low as 35 % Fe, the accompanying sulphur being short in proportion. In addition, the blende and galena contents have increased, though they are still small, and the gangue also is sometimes abnormal now. All this makes the Mount Lyell pyrites a less perfect fuel and flux than it used to be, and the carbonaceous fuel had to be increased, in spite of the

knowledge that such a step, in addition to raising the temperature, would also detrimentally affect the grade of the matte. One consequence has been that the slags have risen in silica contents and fallen in protoxide of iron. They are no longer approximately mono-silicates with 30–32 % SiO_2 and 52 % FeO, but carry 36–39 % SiO_2 and 44 % FeO. From these figures it must not, however, be concluded that the object of the higher SiO_2 is to work off more silicious ore. This point opens up an interesting discussion.

It does not necessarily follow that, in the desire to treat a greater tonnage of silicious ore, we should make more highly silicious slags. In accordance with the pyritic line of reasoning we should only make quantitatively *more slag*, and obtain the fuel and flux for same out of the fixed amount of pyrites by increasing the proportionate quantity of residual sulphide utilized for these purposes. The composition of the slag should be the normal one proper to this idea. Our present position, however, does not allow of this close adherence to the old standard. This included an oxidation factor of 95 % of the iron. We are, however, accomplishing this even when making the present slags with 38 % SiO_2, and we hesitate to carry the oxidation and scorification of the iron any further—first, because the tenor of the matte is quite satisfactory (45 % Cu), and need not be further improved; and second, because of the possible danger of over-oxidation and too small a matte fall. Of these reasons the first-mentioned is the more cogent one. The mere smallness of the matte fall nowadays no longer inspires one with the old fear of metal loss. To prevent over-oxidation we might apply more coke, but this would be bad economy. As a matter of fact, we do have to use a fairly high amount of coke, more than formerly, when there was relatively more pyrites on the charge; but, as already explained, it is for the purpose of keeping up the temperature. When the iron in the Mount Lyell pyrites falls to 35 % the amount of heat generated out of it in the focus is apparently considerably less than it used to be. The influence which the higher coke percentage now has in making the slag more acid is

not contemplated. We acquiesce with this fact because it cannot be circumvented, and because the economics of pyrite smelting does not trouble itself about slag compositions, but turns about a number of points of greater importance. The average copper contents of the ores, of course, also sway the position. In the past we were able to make a 50 % Cu matte with an oxidation factor of only 80 % of the iron ; now 95 % is required for a 45 % Cu matte.

An important occasional constituent of the charge is a clean silica rock, in the shape of barren quartzite, or sandstone, wholly devoid of valuable metals, which is broken off the old silica quarry, contiguous to the limestone quarry. It carries 85 % SiO_2 and over. Before the amalgamation of the old Mount Lyell Company and the North Mount Lyell Company (1903), which made the silicious ore of the later permanently available, this quartzite was the only silicious material smelted together with the Mount Lyell pyrites. The combination was easier to treat than the present one, but the results were not superior to the present, though the coke consumption was less, for the work was then done with a lower column and less blast. This quartzite is now inclined to be too soft in texture (gritty), and is sometimes replaced by conglomerate rock off the mine spoil tips, or out of the mine open cuts, which usually carries a small fraction of a per cent. of copper. At periods coarse conglomerate gravel, from the alluvial deposits in the reduction works valley, has been used. It had to be broken small. for its non-decrepitating character affected the furnace work. The object of employing these barren silica rocks is, as indicated, to give point and incisiveness to the action of the SiO_2 in the charge, whenever the SiO_2 in the ore is not sufficiently free. Nowadays this flux is principally required when the amount of North Mount Lyell ore on charge is already as high as the condition of the furnace will stand, and the matte drops in grade for some otherwise uncontrollable reason. The quartzite is not regularly weighed in as a component of the ordinary furnace charges, but occasionally fed into the furnace in weighed truckfuls. It is instrumental in properly maintaining the "silica skeleton."

Such by-products as flue-dust from the blast-furnaces and the converters, which require treatment, are also not figured as constituents of the ordinary charges. Owing to the great coarseness of all materials, the quantity of flue-dust made by the furnaces is small. It is proposed to machine-sinter it in the future. For the present it is merely pugged. After extensive efforts to agglomerate it cheaply (including bricking, briquetting, fritting in kilns, H. and H. pots, and reverberatory smelting), all of which failed to satisfy, the dust is now simply ground wet in ordinary roller pan mills, together with the cement copper precipitated out of the mine waters, or by itself. About 10 tons of this mixture is divided among the furnaces per day.

Old acid converter linings are dumped into the bins containing the North Mount Lyell ore.

The number of charges of the compositions above detailed, which is put through a furnace per 8-hour shift, in normal running, is 75, counting both sides of the furnace. The average weight, inclusive of flue-dust pug, is 4500 lb. per charge, exclusive of coke.

Such low-grade or first matte as unavoidably falls is smelted as additional material to the ordinary ore charges at times when the converter matte produced by the furnace activity is high. The action restricts itself to a simple dilution of the copper contents of the matte, which the furnace is naturally making out of the ores, with perhaps a little enrichment of the re-charged matte, as the case may be. Accumulations of first matte, which cannot be got rid of in this way, of a tenor from 35 % Cu down, are smelted in charges of the following composition, between limits :—

First matte	1570–1800 lb.
North Mount Lyell ore	800–1200 ,,
Limestone	250 ,,
Silica (quartzite)	100–350 ,,
Slag	50–250 ,,
Coke	160–250 ,,

These matte smeltings need not be specially described, the general procedure being the same as in the ore smelting. The

material is, in certain respects, more tractable than the ore, and a much greater tonnage can be developed. Before the introduction of the present one-stage smelting all our ore was first smelted into a first matte of low grade (15 %), and this then concentrated periodically. This practice, it appears, is still followed at all the other pyrite smelters in the world. If the average of the ores is very low-grade it cannot well be avoided. Where the average copper is the same as ours, or higher, there would seem to be no valid reason for the double smelting. The reason usually given is that the combined slag loss is less than that in the single smelting. Taking the latter at 10 % of the Cu, the recovery in each of the two smeltings would have to be about 95 %, roughly speaking, and this is scarcely likely to be realized in a blast-furnace.

The quantity of first matte that unavoidably forms and has to be re-treated is, however, small—about 10 % of the converter matte made. The furnace is more easily run when producing low-grade matte, nevertheless its formation is not sought. The management of the furnace constantly fluctuates between two extremes—*i.e.*, either too low or too high a grade of matte, depending upon whether there is too little or too much free silica working in the furnace for the other conditions. When there is an excess of silica the matte may go up as high as 60 %, or even 70 % Cu, much transcending a suitable converter grade, and causing high slag losses. Simultaneously, the furnace is liable to slow up very considerably, and, if left to itself, it may come to a standstill altogether.

In view of the necessity of maintaining a proper silica column in the furnace, which it is desirable to create as soon as possible after blowing in, this operation becomes specially interesting. The manner of starting a furnace is peculiar to Mount Lyell. It is carried out with an eye to building up the skeleton as rapidly as possible, so as to curtail the period of the fall of low-grade mattes. These form at the beginning out of the easy charges used in the blowing in. Ordinarily, the formation of a silica column out of the common ore charges would take a number of

days, so the growth of it is hastened by the introduction of free silica in quantity, as a portion of the blowing-in materials, and, for a good start, it is essential that it should be very hot when the ore comes down. The special scheme is as follows:—

The hearth of the furnace is filled up to the level of the tuyeres with carefully laid kindling wood, about $2\frac{1}{2}$ ft. deep. On this is placed a layer of medium heavy split firewood up to the top of the first row of jackets, about 3 ft. thick. On top of this comes coarse, heavy firewood, well spread, up to the top of the second row of jackets, with the knee-bend, about 4 ft. deep, making the total depth of firewood about $9\frac{1}{2}$ ft. Now follow 20 trucks of coke @ 300 lb., total 6000 lb., spread evenly, and then the furnace receives a number of combined slag and silica charges aggregating as follows:—

	Silica. lb.	Slag. lb.	Coke. lb.
2 trucks silica, @ 672 lb.	1344	—	—
6 slag charges, @ 3000 lb. slag, 300 lb. coke	—	18,000	1800
2 trucks silica	1344	—	—
6 slag charges, as above	—	18,000	1800
2 trucks silica	1344	—	—
8 slag charges, as above	—	24,000	2400
2 trucks silica	1344	—	—
2 .. „ if required	1344	—	—
	6720	60,000	6000

In feeding these materials care is taken to spread each constituent evenly, and so that they shall all lie well mixed together. After this preliminary charging the furnace may be ignited at any time—*i.e.*, either immediately or months later. When the moment for blowing-in comes the furnace is fired by means of lighted lumps of oily cotton waste, which are inserted through the two tap-holes, one at each small end of the furnace. A gentle blast is at once used, of from 5 to 7 oz. pressure, until the slag running out of the tap-holes is hot, when it is raised to 9 to 12 oz. The tap-holes are then both closed, and, simultaneously, the channel connecting crucible with sump, underneath the tymp jacket, is opened, and the slag allowed to rise in the sump. The

blast-pressure is increased from now on as the furnace may require. No attempt is made to introduce or cause a heavy pressure in the furnace, and it is permitted to reach its maximum pressure entirely of its own will. It is the blast volume, not the pressure, that does the work; in fact, very successful—*i.e.*, long—campaigns have been run during which the furnace has never reached the usual maximum working pressure, owing to the porous condition of the charge column. This, needless to say, is a very desirable state of things.

As soon as the blast has been put on the furnace a beginning is made with systematic charging. The initial ore charge is of the following composition:—

Mount Lyell pyrites	2240 lb.
North Mount Lyell ore	1000 ,,
Silica rock	200 ,,
Limestone	250 ,,
Slag	550 ,,
Coke	250 ,,

The grade of the matte is tested as soon as it comes down, and, from now on, changes in charge are made entirely in conformity with the furnace action. When the matte has reached converter grade the silica rock is taken off, and when the furnace has become thoroughly hot, and appears set in action, the coke is reduced to 200 lb. This particular change then forms the basis from which all others are developed thereafter.

It will be instructive to follow the furnace action as regards oxidation factor, slag formation, etc., through its course by means of a simple calculation at the hand of a definite run. The charge changes so frequently each day, even on one furnace, that the opportunity does not often present itself to collect a set of data which are representative of a continuous run of any length, but the annexed schedule of figures (p. 116) pertains to a recent full day's run, on the same charge, without change, and permits of drawing conclusions without the danger of their perhaps being based on interfering factors. In other words, the results arrived at will not be mere averages from several varying sets of conditions, but will be representative of uniform, standard work.

May 11th, 1915. **Table of Charge Smelted and Results.** Furnace No. 5.

Material.	Gross Weight.	H_2O	Dry Weight.	Cu		SiO_2		Fe		BaO		Al_2O_3		Zn		Pb		CaO + MgO		S
	lb.	%	lb.	%	lb.	%	lb.	%	lb.	%	lb.	%	lb.	%	lb.	%	lb.	%	lb.	%
M. L. pyrites	2240	2	2195	0.48	10.5	5.4	118.5	39.4	864.8	2.4	52.7	2.4	52.7	1.7	37.3	1.2	26.3	—	—	46.0
N. M. L. ore	1500	3	1455	6.97	101.4	66.0	960.3	6.9	100.4	2.0	29.1	9.5	138.2	—	—	—	—	—	—	7.5
Slag (conv.)	400	—	400	2.5	10.0	29.0	116.0	46.7	186.8	—	—	5.5	22.0	—	—	—	—	—	—	—
Limestone	250	—	250	—	—	6.5	16.3	1.5	3.8	—	—	2.0	5.0	—	—	—	—	49.0	122.5	—
Totals	4390	—	4300		121.9		1211.1		1155.8		81.8		217.9		37.3		26.3		122.5	
Coke ..	250	10	225																	

No. of charges per 24 hours.—70 + 73 + 75 = 218.

No. of tons (2240 lb.) per 24 hours.—

Mount Lyell pyrites	..	218
North Mount Lyell ore	..	146.1—364.1 total ore
Limestone . ..	..	24.3
Slag	..	38.8
Coke	..	24.3
Pugged flue-dust ..	..	16.2
Silica rock ..	..	3.3
		471.0 total burden

Matte fall, 148 pots = 25.6 tons.

Matte assay, from 50 % Cu to 40 % Cu; average for the day, 43 % Cu (by shift bosses' determinations).

Slag assay, 0.4 % Cu for the day.

Slag analysis, average for the day:—

		%
SiO_2	.. .. ..	38.6
FeO	.. .. ..	44.9
CaO	.. .. ..	3.8
Al_2O_3	.. .. ..	7.2
BaO	.. .. ..	2.4
ZnO	.. .. ..	0.7
		97.6

Typical analyses may be gathered from this table. The average composition of the two ores together is of interest :—

	%
Cu	3.07
SiO_2	29.50
Fe	26.50
Al_2O_3	5.24
$BaSO_4$	3.43
Zn	1.02
Pb	0.72
S	30.80
	100.28

The Cu on charge happened to be rather high, and consequently the average concentration ratio is not as high as usual. A few trucks of silica rock and some flue-dust pug were charged during the day, but need not be taken into account. The calculation is only a rough one, in any case. Therefore the ash in coke, the flue-dust loss, and refinements about the constitution of the slag are also not considered.

In looking about for a starting-point to base the estimation of the oxidation factor on, in the absence of the actual weight of the slag and the matte, we may get at it by first computing the weight of the slag made from the SiO_2 percentage of the slag and the weight of SiO_2 on charge, and then considering the FeO. Thus the total weight of the slag is $1211.1 \div .386 = 3137$. Similarly, the weight of Fe in the slag made is $3137 \times .449 \times \frac{7}{9} = 1095.51$. Now, the Fe in the old slag charged and in the limestone, both of which were already oxidized, is $186.8 + 3.8 = 190.6$. This has to be deducted from the original total Fe on charge, in order to ascertain what weight of Fe was available for oxidation by the furnace—viz., $1155.8 - 190.6 = 965.2$ Fe. The same 190.6 Fe must also be deducted from the Fe in the slag made, to arrive at the Fe actually oxidized by the furnace, thus : $1095.51 - 190.6 = 904.91$. To obtain the oxidation factor we have to compare the two figures for amount of Fe, and thus we get $904.91 \div 965.2 =$

.937, as representing the percentage of iron in the ores which the furnace action has oxidized and scorified. This is the average oxidation factor for the whole day, and for a matte tenor of 43 % Cu only.

It is, however, more interesting to follow the furnace activity through at the hand of the matte assay, and without knowing the slag analysis, as this puts one into closer touch with the actual readjustments of the chemical constituents which takes place within the furnace. For this purpose it is necessary to have a starting point, and at first sight there does not appear to be one. Some constant seems to be wanted which goes through the furnace unchanged in weight. As we are not supposed to know the slag analysis, the computation of the oxidation factor from the matte tenor only appears barred. Fortunately, however, there is one peculiarity, common to all mattes, which can help us out of the difficulty. This is the striking fact that all copper mattes, for some singular reason, show a practically constant percentage of sulphur—namely, about 25 %. The constituents found by chemical analysis, moreover, never add up to 100 % but the ordinary ones, like Cu, Fe, and S only run to about 95 % in the analysis. Both these figures are only claimed to be approximately representative. It is a matter of choice which variation one takes out of the slight departures which they both show from those here employed. For our present purpose the compositions of the three mattes to be dealt with, of which we have actually only determined the copper tenor, may be set down as approximately the following :—

	%	%	%
Cu	50	43	40
Fe	20	27	30
S	25	25	25
	—	—	—
	95	95	95

With the aid of these rough and ready matte compositions we can calculate the oxidation factor on the iron, and can also construct the analysis of the slag that forms, without knowing its

weight, and will get a mathematical insight into the rearrangement of the substances in the focus.

It is necessary, first, to make an allowance for the Cu which does not find its way into the matte, but is lost in the slag, in order that we may not overstate the weight of Cu in the matte made. For this purpose we will have to use the Cu assay of the slag and the weight of the latter, which can be ascertained, as above, with sufficient accuracy, from the knowledge we have of the usual SiO_2 contents of the slag—viz., 38.5 %. We thus have 1211.1 ÷ .385 = 3146 slag. We might also use the CaO on charge, the contents of the slag being usually about 4 %: 122.5 ÷ .04 = 3062 slag; but, owing to smallness of figures, this result is apt to be further from the truth.

We then have, for the Cu wasted in slag, 3146 × .004 = 12.58 Cu. This indicates a loss of 12.58 ÷ 121.9 = 10.3 % of the total Cu through the slag, and leaves 121.9 − 12.58 = 109.32 Cu for the matte.

We will first calculate the oxidation factor for the 50 % Cu matte. The weight of such matte formed is derivable from 109.32 ÷ .50 = 218.64 matte. This contains 218.64 × 0.20 = 43.73 Fe. This Fe has to be subtracted from the 965.2 Fe found, as above (186.8 + 3.8 = 190.6; and 1155.8 − 190.6 = 965.2), to be the weight of Fe which is available in the ores for oxidation. The result is 965.2 − 43.73 = 921.47 Fe, for the quantity which has not gone into the matte, but has been oxidized and scorified. We finally compare the figures and find that 921.47 ÷ 965.2 = .955 is the oxidation factor of the furnace, when making this high-grade matte of 50 % Cu.

In the case of the average matte, of 43 % Cu, the calculation is as follows:—Weight of such matte, 109.32 ÷ 0.43 = 254.23. Weight of Fe in same, 254.23 × 0.27 = 68.64 Fe. This Fe, escaping oxidation, leaves 965.2 − 68.64 = 896.56 Fe for scorification, and the oxidation factor is 896.56 ÷ 965.2 = .929 Fe. This figure agrees sufficiently well with the factor calculated above, from the slag analysis, as .937 Fe.

For the 40 % Cu matte the oxidation factor similarly is found to be .915 Fe.

It will be remarked that even the lowest of these factors is very high. It will also be patent, on reflection, that the expression of the oxidizing power of a furnace by means of percentage figures, like 95 %, &c., though admissible for works purposes, is not really suitable for the comparison of the quality of the smelting done in different places on different ores. For this purpose the weight of the amount of iron scorified in the unit of time is the only thing to use. For our own 43 % Cu matte, in the example before us, the oxidation took place at the rate of 136 lb. Fe per min., and, for a 50 % Cu matte, if made all day long, it would have been 140 lb. Fe per min. The mere percentage factor is not sufficiently definite. For instance, 95 % of the Fe is materially less on a pyrites with only 35 % Fe than on a pyrites carrying 45 % Fe. This fact is expressive of the very departure from past practice which is now imposed upon us at Mount Lyell by nature, and explains our greater coke consumption.

It will be noticed that no attention is being paid to the combustion of the sulphur. The reason will be plain from what has been said above respecting this element (p. 85). I may take the opportunity also to say that it will be admitted that the common practice of trying to convey the oxidizing power of a pyritic furnace in terms of the percentage of desulphurization of the ores is inexact, and mis-states the case. The elimination of the sulphur is not a guarantee that the furnace has derived any thermal benefit from this element. Furthermore, the percentage depends altogether upon the original total quantity of sulphur charged. If the original amount of sulphur present is small, which it frequently is in pyritic work (in distinction from pyrite smelting), the percentage of desulphurization may figure out quite high, and the bessemerizing or calorific effect of the furnace mixture be regarded as very good, when, as a matter of fact, it is not so. The actual weight of the sulphur eliminated per unit of time is the proper criterion, not the percentage. However, as the prime object of the treatment is to oxidize, and the sulphur, for the greater part, eludes the blast and is not subject to this action, it is undeniable that the associated element, iron, affords

a much more satisfactory standard for gauging the performance of the furnace or of the metallurgist.

There now remains the instructive task of arithmetically constructing the slag composition from the data to hand, and of comparing the result with the analysis of the actual slag, as it ran out of the furnace. We only knew the assay of the matte, to begin with, but have calculated the Fe oxidized and scorified. In addition, we have the quantities of the other constituents which entered the slag, given in the table on p. 116. In the case of the Zn, which does not all go into the slag, it is sufficiently close to assume that one-half is slagged as ZnO. The Pb may all be regarded as sublimed, none entering the slag. The $BaSO_4$, we believe, is broken up, and the barium is, at least for the greater part, present in the slag as oxide, BaO, not as BaS. The sulphur content of the slags is always very low, and, as it has to be allocated between Zn, Fe, Cu, and Ba, there is no essential amount of S left for the last-mentioned. In the blast-furnace it is likely that the $BaSO_4$ is entirely decomposed by the SiO_2 in the presence of the iron sulphide and the neutral gases, with the formation of barium silicate.

We can set up the following list of slag-builders :—

SiO_2		1211.1
FeO, from the 896.56 Fe oxidized	1152.7	
,, ,, ,, 190.6 ,, in old slag and limestone	245.0	
		1397.7
CaO		122.5
Al_2O_3		217.9
BaO		81.8
ZnO, from 37.3 ÷ 2 = 18.65 Zn		23.2
Cu, as calculated		12.6
Total		3066.8

We know from daily experience that the collective weights of these constituents aggregate 98 % in the analysis, consequently the total weight of the slag is 3066.8 ÷ .98 = 3129.

It is now possible to calculate the percentage composition of the slag, the results being as follows. For comparison the analysis of the slag, as actually made on the collective sample for the 24 hours, is also set down.

	Calculated slag analysis. %		Actual slag analysis. %
SiO_2 ..	$1211.1 \div 3129 = 38.70$	..	38.6
FeO ..	$1397.7 \div 3129 = 44.67$	..	44.9
CaO ..	$122.5 \div 3129 = 3.92$	..	3.8
Al_2O_3 ..	$217.9 \div 3129 = 6.96$	..	7.2
BaO ..	$81.8 \div 3129 = 2.61$	..	2.4
ZnO ..	$23.2 \div 3129 = 0.74$	..	0.7
Cu	$12.6 \div 3129 = 0.40$	..	0.4
	98.00	..	98.0

The coincidence of these figures is striking. A better agreement could not be desired, and it is, in essence, a confirmation of the opinions here advanced as to the inner reactions of the pyrite furnace. The respective phenomena are all simple enough, but they require a readjustment of one's views on the subject of the chemical and physical work being done in the furnace, under our peculiar conditions, which it takes observation and thought to bring into clearness. The above example will, I hope, be found useful to point the interchange of substances involved, as it is regarded at Mount Lyell.

It will doubtless have been noticed that the base metal, on account of which the smelting is done—viz., the copper—has scarcely been drawn into the discussion, and that the two precious metals which accompany it—viz., silver and gold—have not even been mentioned. As to the latter, the reason is well known—their great affinity for copper and sulphur simply makes them follow where the copper leads. As regards the red metal itself, the explanation is that it is perfectly safe to assume that, as long as there is a sufficiency of sulphide of iron present, the copper is fully protected by its great affinity for sulphur, and plays no *rôle* whatever in the metallurgical reactions. It passes through the

furnace like an indifferent substance, in company with the iron sulphide, and, while the latter is subject to progressive disintegration into its elements, and finally an almost complete oxidation, the copper sulphide remains chemically intact throughout. Roughly speaking, the residual sulphide which reaches the focus may be regarded as having the following empirical composition : $m\ Fe_5\ S_4 + n\ Cu_2S$ (where the $Fe_5\ S_4$ is only approximate and might be any other similar formula, Fe_4S_3, etc., or even Fe_2S). In this combination m and n represent the *weight* of the respective combination before which they stand. The position of Cu in the furnace is so secure that even 95 % and more of m may be completely burnt in the focus, and the iron slagged, without any portion of n being touched by the blast and the silica. Doubtless there is a limit to this security—*i.e.*, there must be a minimum value for m, where it is insufficient to protect $n\ Cu_2S$; but this state of affairs has not even remotely been approached in pyrite smelting. No scorification of copper has ever been noticed, at Mount Lyell at least. One may also be allowed to infer, from the analogy of the second stage of the converting operation, where only Cu_2S is being blown, that, even without the iron sulphide, the scorification of the copper would be difficult. However, it would very likely take place in the blast-furnace, whereas it does not actually, to any extent, do so in the converter. The respective furnace action, however, would be self-extinctive, so to say, for the metal is not a good fuel, and a blast-furnace burning up copper would soon, automatically, put an end to its own ruinous career.

In connection with the above-mentioned rough formula for the residual sulphide ($m\ Fe_5\ S_4 + n\ Cu_2S$), it may be pointed out that the composition of the residual sulphide is necessarily also that of the matte which falls, since the matte is only that portion of it which has escaped the influence of the blast and silica. Furthermore, it follows that the combination of Fe, Cu, and S which does the work in the focus of the pyrite furnace is also the very one which does it in the converting operation, to which the matte is subsequently subjected. The recognition of these facts

leads to the belief that the wholly cokeless pyrite smelting ought not to be beyond the reach of practical politics in metallurgy, and will doubtless be accomplished some day, even without a greater utilization of the elusive sulphur. The converting operation receives the benefit of the superheat in the matte discharged from the blast-furnace, and it may, roughly speaking, be said that the coke now used supplies this superheat. On the other hand, the utilization of heat in the pyrite furnace is not good, and it may be that, if this waste is properly minimized by a more efficient furnace construction, the coke can be spared.

MINING METHODS AT MOUNT LYELL.

By R. M. Murray.

The mining methods employed in the Mount Lyell Mining and Railway Company's mines are, perhaps, less known to the mining world than those of other important districts, due to the geographical remoteness and isolation of the Mount Lyell field, but they present many features which are unusual, and should be of interest to the mining engineer.

Typical mining is carried out on an open-stope system, details of which are the result of local experience and evolution, and of which the outstanding feature is that the ground is so worked as to be largely self-supporting by virtue of domical form given to the stope roofs. It is locally termed "rill" stoping, but differs widely from what is generally understood by that designation, chiefly in that the "rill" or slope is developed in every direction from a central focus, and not merely in directions following the strike of a lode or ore-body whose longitudinal dimensions are very much greater than its width. Such terms as "dome stoping" or "arched-back stoping" would be more appropriate and descriptive.

The company's active operations at the present time extend to the Mount Lyell and North Lyell mines, and, though the fundamental principles of stoping methods are the same in each, the dissimilarity of the ore-bodies, both in nature and mode of occurrence, necessitates considerable variation as to details to meet the respective conditions. The Mount Lyell pyritic body lends itself to the adoption of the system in its simplest and almost ideal form, and a description of stoping in this mine will serve best to illustrate it, while later reference to details of practice in the North Lyell mine will indicate how it may be varied to meet more difficult circumstances.

[This paper is separately bound, and may be, if so desired, detached complete from this number.]

MOUNT LYELL MINE.

The ore-body in the Mount Lyell mine is a huge mass of cupriferous iron pyrites, irregularly elliptical in shape in horizontal section, and, with the exception of a few schist intrusions, is remarkably homogeneous and solid in character, though denser and harder on the foot-wall than on the hanging-wall side. The walls are well defined, the foot-wall being formed by the contact conglomerate, although that rock is almost invariably separated from the pyrites by a band of schist of varying thickness; it is strong and has a marked and regular underlay, and forms an excellent "skewback" for the stope arches. The hanging-wall is of the typical grey micaceous schist of the district, which flakes off fairly readily, and rarely has much strength; it is irregular in underlay, varying from nearly horizontal to nearly vertical, necessitating that the stoping method adopted must provide for its being inherently weak and make special provision for the excavation of the ore below it.

The ore-body has a general south-westerly underlay of 26° from the vertical between the surface and No. 7 level, and thence downwards it flattens very considerably. From the outcrop down to No. 5 level, a vertical distance of 350 ft., it has been extracted almost completely by open-cutting, except in its two extreme ends. The portion below No. 5 level is being worked for a depth of 250 ft. by underground mining. Its size varies considerably at each level, chiefly due to irregularities in the hanging-wall above referred to. The original body was largest at No. 4 level, where its dimensions were 660 ft. long by 270 ft. wide. Between this and No. 5 level, a vertical distance of 150 ft., a considerable shortening takes place in the northern end, and the dimensions of the lower levels are :—No. 5, 510 ft. by 210 ft.; No. 6, 510 ft. by 270 ft.; No. 7, 400 ft. by 300 ft.; No. 8, 350 ft. by 200 ft. A marked diminution takes place between Nos. 7 and 8 levels, and below the latter no exploratory work beyond diamond-drilling has been carried out, No. 9 level having only just been reached by the main shaft.

The conditions under which underground extraction was inaugurated presented unusual opportunities for attacking the ore-body by some completely pre-determined method, in that

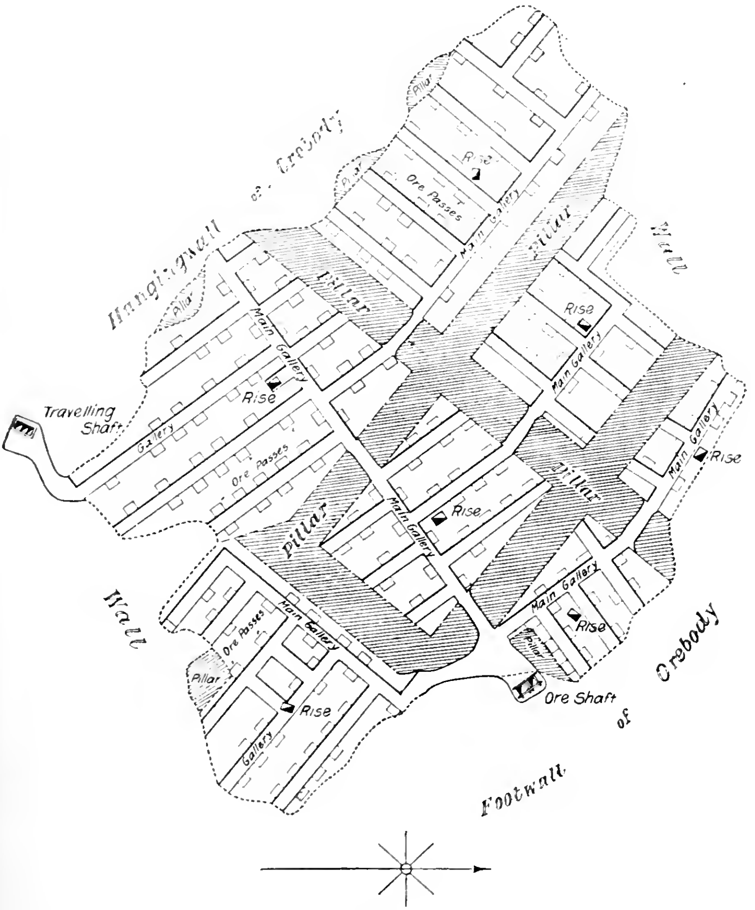


FIG. 1.
PALN OF ORE-BODY, MOUNT LYELL MINE, SHEWING LAYOUT OF STOPES, GALLERIES, ETC.
(Scale—100 ft. to 1 in.)

its size and shape had been already ascertained by contour drives at Nos. 5, 6, and 8 levels, and were also partly known at No. 7 level, while the body itself below No. 5 level was absolutely intact.

As a matter of fact, the complete scheme of working that has been carried out was designed in the office before a shot was fired in the mine, and the original design has been given effect to practically without any modification.

The method adopted is to cut up the body at each level into suitable stoping sections, divided by pillars of solid ore, 25 ft. thick, which are continuous in vertical planes throughout the mine, there being two transverse pillars, 100 ft. apart, and two longitudinal ones also 75 ft. apart, as shown in Fig. 1. These pillars develop stoping-areas, which they either circumscribe entirely, or which are bounded partly by them and partly by the natural walls.

The area of each stope being thus outlined, sill-floor stoping in each section is commenced from some convenient point of attack, and the ore beaten out for a height of about 12 ft., the back being left flat and temporarily supported by stacks where necessary; these are, however, not extensively required. Where the underlay is considerable, small isolated pillars are left under the hanging-wall to provide strength in case of erratic flattening.

When the sill-floor stope has been well advanced, a rise is put up from a central position to connect with the mullock-pass at the level above, and the construction of the galleries, or travelling-ways, is commenced. These are laid out with special regard to the placing of ore-passes to adequately serve the overhead stope, and usually comprise one main gallery leading direct to the ore-shaft, with cross galleries at right angles to it, and 21 ft. apart, centre to centre, extending to the walls, or pillars, as the case may be. These galleries are constructed of strong square-set timber, forming cubical cells, 7 ft. centre to centre, and 8 ft. 6 in. high, the legs being specially large—often from 2 ft. to 2 ft. 6 in. in diameter—in order to provide good seating for the caps and struts and afford stability against the flow of mullock during the initial filling operations. The sills of these sets are laid down on heavy round lagging, 15 ft. long and 8 in. to 10 in. in diameter, to facilitate picking up the level subsequently. The legs are temporarily stayed in position by raking

props, and also by crown tomming to the back. The sets are laced up on the outer sides with round lagging or sawn sloven from the timber yard, and covered over on top with a double layer of heavy round lagging laid transversely upon stringers placed longitudinally along the struts. The floors of the panels formed between the cross galleries are also covered with heavy round lagging, laid breaking joint, and the portion of the stope around the central mullock rise is then ready to receive filling.

To facilitate this the bottom of the rise is given a wide bell-mouth when rising is commenced, so that, when filling is first passed down, it can be spread over a considerable area. The sides and tops of galleries having been lined with old coke bags to prevent fine material escaping through them, mullock is run in and allowed to fill as much of the nearest panels as possible, and so support and cover the adjoining gallery timbers, and everything is then in readiness for the first overhead stope to be commenced. This is undertaken from the rise, working outwards and downwards in all directions, enlarging the original bell-mouth into a small domical chamber. Mullock is then again run in, covering further portions of the sill-floors and timbers, and forming a cone below the rise from which to begin further breaking operations. When several small stopes have been taken off in this manner a fairly large, dome-shaped chamber containing a corresponding cone of filling has been formed, and the stope is then well opened up for vigorous ore-breaking, the domed roof eventually covering the whole of the sill-floor area, and having for its abutments the pillars or foot-wall, as shown in (Plate I), in which the development of several stopes is indicated by means of contours.

The multiplicity and placing of ore-passes are a special feature of the system, and their construction is taken in hand immediately prior to each mullocking. They are built of 10-in. round or split timber joggled at ends, their size being 6 ft. by 4 ft. They are placed on each side of the galleries at such distances that their centres are about 15 ft. apart horizontally, the close spacing reducing the handling of the ore in the stope to a minimum, and

also rendering the wear on the timbers very much less than would be the case were all the ore broken handled into fewer passes. As the height of the stope increases, however, and the slope of the filling becomes greater, the ore can be conveniently handled over greater distances, and some of the passes are then discarded, being allowed to fill completely with mullock. The ones discarded are naturally those situated towards the central portion of the stope, those on the periphery being rigidly maintained, although it is an essential feature of the system that the ore is not made to gravitate to the bottom of the slopes, as is generally the case with "rill" systems, but can be readily shovelled or raked direct into a pass, whatever its position in the stope may be. It may be mentioned that some of the larger stopes in the mine are served with as many as fifty ore-passes. Prior to each mullocking each pass is built up to the back, against which it is wedged for support, and the logs are further held in position by vertical timbers placed against the sides and ends, those on the downhill side being strutted with raking props to withstand the force of the mullock as it is being run in. As in the case of the galleries, the outsides of the passes are lined with old bags to prevent the passage of fine material between the logs. Passes contiguous to pillars and over the central portion of stope are kept vertical, and those adjacent to walls are pulled over to conform to the underlay of same, except in cases where the hanging-wall is very flat, when they are discarded. The vertical ones do not require inside lining, but where there is any considerable underlay the lower side and ends are sheathed with 8-in. by 2-in. planking to prevent the wearing away of the logs. Immediately below the mullock rise a special pass is constructed for transferring mullock to the level below. This is sheathed with 8-in. by 2-in. planking, and has a manway partitioned off. Manways, 3 ft. by 3 ft., are also built in each corner of the stope. Where necessary, individual passes are built up to the back as stoping proceeds, to serve temporarily as stacks for supporting weak or heavy portions of back. All timber used for pass-building is thrown down the mullock rise from the level above, reducing the handling cost to a minimum.

Mullock for filling purposes is obtained from the open-cut, where it is broken in the ordinary course of overburden removal, and, if not utilized in this way, would be tipped to spoil, as was the case prior to the commencement of underground mining. It is delivered direct from waggons into each of the numerous passes leading to the stopes, as required, water being simultaneously led into the pass, so that when it reaches its destination it has a consistency similar to that of rather-wet concrete. The flow of mullock is controlled by a chute door at the level immediately above the stope to which it is proceeding. Under favourable conditions—that is to say, after a stope has been well opened up and the requisite slope of back attained—about 75 % of the mullock required to fill it will run into position by gravity alone, the only labour expended being that of one and sometimes two men per shift controlling the overhead chute, and, into some of the larger stopes, a quantity of 5000 yd. (broken) is sometimes delivered at one filling. When the stage is reached that the mullock ceases to distribute itself by gravity, a gang of mullockers proceeds with the work of leading it to the remote corners, which is effected by laying timber slides of 8 in. by 2 in. planks from the pass to the place to be filled. The wet mullock travels readily along these slides, even when laid with very slight inclination, with but little assistance from men stationed with shovels at intervals along their length. Over the central portion of stope the filling is allowed to reach to within a foot or two of the back, leaving sufficient space to permit of the next layer of ore to be shot down and fall freely. Under the hanging-walls and near the pillars it is allowed to run right to the back, and so afford direct support. The character of the filling is such that, when mixed with water, it soon sets very hard, requiring picking to break it, and it subsides very little after being put in. In fact, in cases where it has been necessary to make excavations in it, it has been found to stand readily with high vertical and even overhanging faces. For this reason it has been considered unnecessary to place lagging against the pillars, as no difficulty is anticipated due to "running" when these are being extracted. When mullocking

is nearing completion fine material only is run in, in order to fill up the interstices between the large fragments, and so form an even surface upon which to break down the ore. This surface can be readily scraped down when the stope is being cleaned up prior to the next mullocking, thus ensuring that none of the broken ore is lost.

When breaking and mullocking operations are carried out alternately, as above described, all production from the stope necessarily ceases during the latter operation, and this, in the case of large stopes, means considerable interference with the total mine output. This interruption, however, can be overcome by so working the stope that only half of it is beaten out at a time, then erecting a barricade at the rise which will direct the filling to the depleted portion while the second half of the stope is being worked, and this is now the ruling practice in the larger stopes of the mine.

The highest portion of the stope being always at the mullock rise, it is here that the overhead level is first reached. The method of "picking up" is to flatten off the apex of the cone of filling at a convenient height for one set of timber—usually about 6 ft. below the overhead lagging—and so form a base on which to erect the 5-ft. by 5-ft. picking-up sets. The levelled-off area is covered with heavy round lagging, which serves to spread the pressure, and on this the sill timbers of the sets are laid. The filling forms a firm and almost unyielding foundation, being even more than ordinarily consolidated in this locality, due to the ramming action of the mullock during the running of the pass. When the sills have been laid immediately below and around the rise the collar of latter is broken away, and the overhead lagging caught up on square sets erected on the sills, and securely packed and wedged up, stringers being used on top of the sets where necessary. As each succeeding stope is taken off the picked-up area is increased by the addition of one or more rows of sets around those already in position, each successive mullocking increasing the area of the flat base on which they are erected. By means of timber slides the mullock required for the lower

portions of the stope can be directed from the chute at the overhead level through the picking-up sets to the summit of the rill, and will flow in this way, by gravity alone, over a radius of 20 ft. (horizontal) round the pass in every direction. When the picked-up area extends beyond this limit the flow needs to be assisted on the slides by a certain amount of shovelling, and the filling is handled in this manner until a radius of 30 ft. from the pass has been attained, it being only the portion of the stope beyond this limit that requires its filling to be transported by trucking. As the picking-up sets are stood as many as possible are packed tight with filling, only those required to form passage-ways for the mullock slides being left open. The ore-face round the timber is always kept steep, being the upper limit of the sloping sides of the stope, and this ensures that there are no thin plates or crusts forming the roof of stope and floor of overhead level, and consequently no weakness or danger of collapse of latter on this account. The portion of the level picked up in this manner gradually subsides to a small extent, the subsidence, however, rarely exceeding more than about 12 in., and is always uniform, very little disturbance of the level timbering being noticeable.

In the general work of ore-breaking 3-in. piston machine-drills are used, each being operated by one man, and two machines working in close proximity, so that the drillsmen can assist each other in lifting, etc., the shovellers being also available for this purpose. Commencing at the rise, and working outwards and downwards, the holes drilled are roughly parallel to the slope of the filling, and are all consequently down or "water" holes, which conduces to leaving a clean back, and also minimizes dust. A water-jet is used with each machine. The ore usually breaks big, and is reduced to a convenient size for spalling by "popping" with hammer-drills, and sometimes by "blistering." When approaching any flat hanging-wall the work is planned as far as possible to cause the ore to settle on the filling in large masses, when it can be worked out in safety, thus avoiding the necessity of working under "shells" of ore too thin to be self-supporting.

The layer of ore taken out in each "break" is usually 6 ft. or 8 ft. thick, measured normally to the slope of the filling, and is generally taken out actually in two layers, the machines going over the whole of the back twice. In any place where the roof appears to need special support stacks are erected, but it is rarely that many are required, and, as already mentioned, ore-passes are frequently built up to the back as substitutes.

The leading features of the system are shown in Fig. 2 which is a vertical section through two adjoining stopes, one on the foot-wall side and the other on the hanging-wall side of the ore-body, separated by a pillar. The section as drawn is based upon an actual working section, but certain modifications have been made to fully illustrate various details already described. The hanging-wall stope is shown as it would appear just after ore-breaking has been finished over its entire area, the stope being ready for pass building and for the running in of filling. The foot-wall stope is shown as it would appear immediately after filling operations have been completed, and when ore-breaking is about to be re-commenced. The central portion of this stope is shown picking up the overhead level. In each stope the mullock-pass is shown logged up through the filling, and its continuation to further levels above and below the stope is indicated.

The principal advantages claimed for the method are—

(1) Strength of the stope roofs, due to their domical form and to the avoidance of opening up any great area of any horizontal layer, or head.

(2) Economy in handling ore in the stope.

(3) Economy in filling stope.

(4) Economy of timber for support, the few stacks required for this purpose being almost invariably recovered.

(5) Accessibility of all portions of the back for inspection, barring, &c., any part being easily reached horizontally from some point on the filling.

(6) The general advantages arising from the boring of all down holes, as well as the important one of minimizing dust.

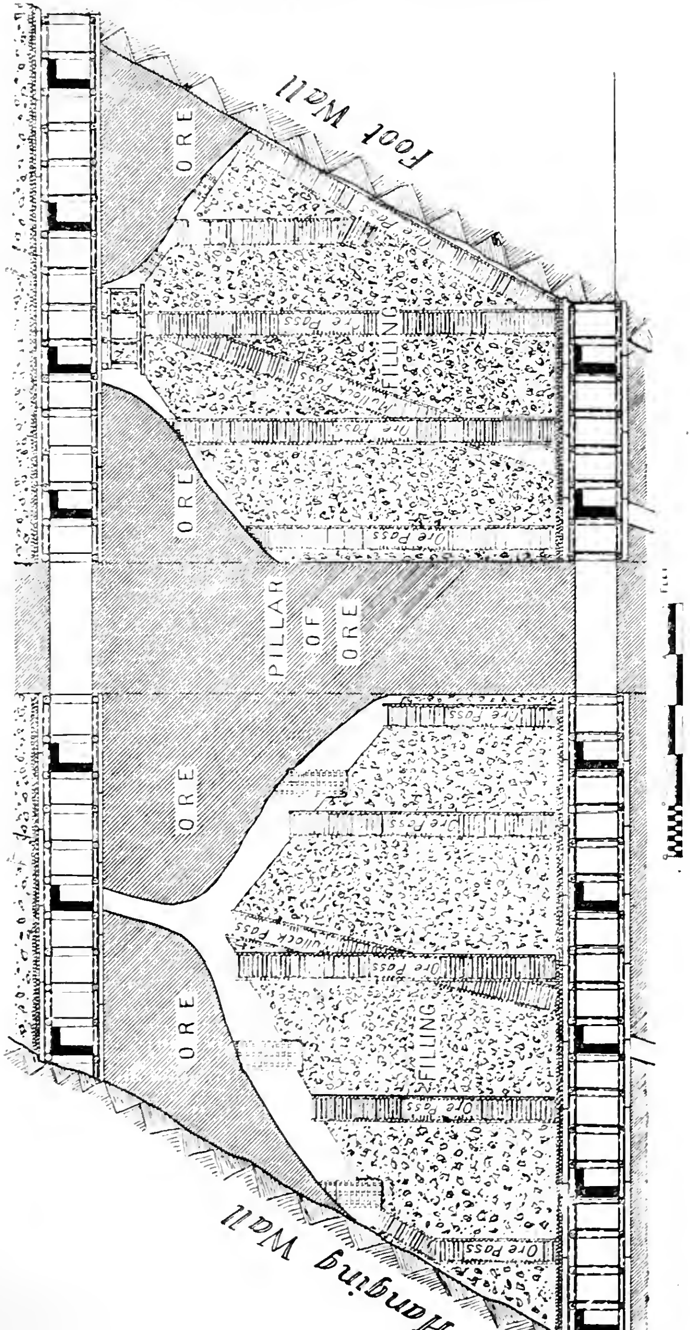


FIG. 2.
VERTICAL SECTION ALONG CENTRE-LINE OF STOPES.

The following particulars should serve to justify these claims. The average amount of ore broken per shift per machine is 25 tons; the average amount of ore spalled and shovelled into passes per man per shift is 25 tons, and as much as 100 tons have been handled by one man in a shift under favourable conditions. The average amount of ore won per man per shift, counting all hands engaged in a stope, is $10\frac{1}{2}$ tons [The specific gravity of the ore is high, slightly less than 8 cub. ft. (solid measurement) constituting 1 ton].

The actual working costs of a typical stope, taken over a period of six months, are as follows:—

	Per ton (dry). s.	d.
Drilling, spalling, and handling ore into chutes ..	1	5
Timbering	0	3
Mullocking	0	3
Trucking	0	$4\frac{1}{2}$
Winding	0	$2\frac{1}{2}$
Steel sharpening	0	1
Explosives	0	3
Stores	0	6
Timber	0	5
Power	0	3
Miscellaneous (repairs, etc.)	0	10
Supervision	0	5
Total cost delivered into service bin at incl. shaft	5	3

The contract price for breaking the ore and handling it into the chutes varies from 1s. 6d. to 2s. per ton, the contractors supplying their own explosives and stores, and paying for steel sharpening, but the company providing power, timber, etc. The average contractors' earnings per man are 16s. per shift. In the above statement of costs power is put down at 3d. per ton. Prior to the installation of the hydro-electric scheme this cost was 9d. per ton, and when the air-compressing plant is completely electrified a further reduction will be made.

NORTH LYELL MINE.

At the North Lyell mine the same general methods are adopted, but certain modifications are necessary to meet particular conditions obtaining in this mine. The ore is siliceous in nature, comprising quartzites and schists carrying various copper minerals —principally bornite, chalcopyrite, and chalcocite—and is much less dense than the pyrites of the Mount Lyell mine, about $12\frac{1}{2}$ cub. ft. (solid measurement) constituting 1 ton. The ore-bodies are erratic in shape, and vary largely in size, and do not permit of being opened up on a pre-determined scheme as in the case of the Mount Lyell ore-body, as their horizontal limits can only be defined by actual sill-floor stoping as guided by assay value, well-defined walls being rarely encountered.

The quartzitic ore is far more prevalent and important than the schisty type, and is generally intensely hard and stands well, the schisty ore being weaker, and often requiring to be taken out on timber as well as filling. When the nature of the ground permits, the sill-floors are taken out and galleries stood as at the Mount Lyell mine, but occasionally the whole floor requires to be timbered with square sets, which are ultimately filled in, except those which are laced off and kept open to serve as travelling-ways, etc. In such cases, however, it is frequently found that, as overhead stoping proceeds and the domed shape of the roof is acquired, the timber can be dispensed with, the natural strength of the ground being then sufficient to render it self-supporting. Even, however, when square-set timber is required throughout the whole vertical height of a stope, the domical form is still adhered to, both with a view to securing the advantage of whatever natural strength the ground has, and also to facilitate filling, which is carried out just as in the case of an open stope. Each new floor is opened up round the central mullock-rise, and the lower floors are advanced successively a few sets at a time after each mullocking.

When a stope is being worked on the "open" system, and any bad ground is encountered which cannot be controlled by stacks,

square-set timbering is adopted over the affected portion, the sets being stepped down the rill on benches cut in the filling. Such timbering, however, is generally only necessary over short vertical distances, being discontinued when the ground becomes strong again, in which case a considerable portion of the timber is recovered.

Owing to the irregular and erratic shape of the ore-bodies, it is rarely that symmetrically-shaped stopes are developed, and there are many instances of excavations whose length is very much greater than their breadth. In such cases more than one mullock-pass is required, the maximum distance apart at which these are situated being about 100 ft. A central pass is maintained in the highest portion of the stope, and subsidiary passes are placed at intervals down the slopes along the axis. The main feature to be observed in this arrangement, however, is that the stope be not allowed to develop into several domed stopes, each around its own pass, with a heavy pendular mass of ore separating each from its neighbour. For the proper working of the stope the crown of the arch must be at the central pass, and the slopes continuous from that point to the lowest limits in all directions, the subsidiary rises merely entering the stope at points along such slopes for the purpose of minimizing the distance which the filling material has to traverse to fill the remote portions of the excavation. Such an arrangement is illustrated in Plate II., where the slope and configuration of the back of a large stope is shown by means of contours, the central mullock-rise being nearly 100 ft. above the bottom of the rill along which intermediate rises are situated.

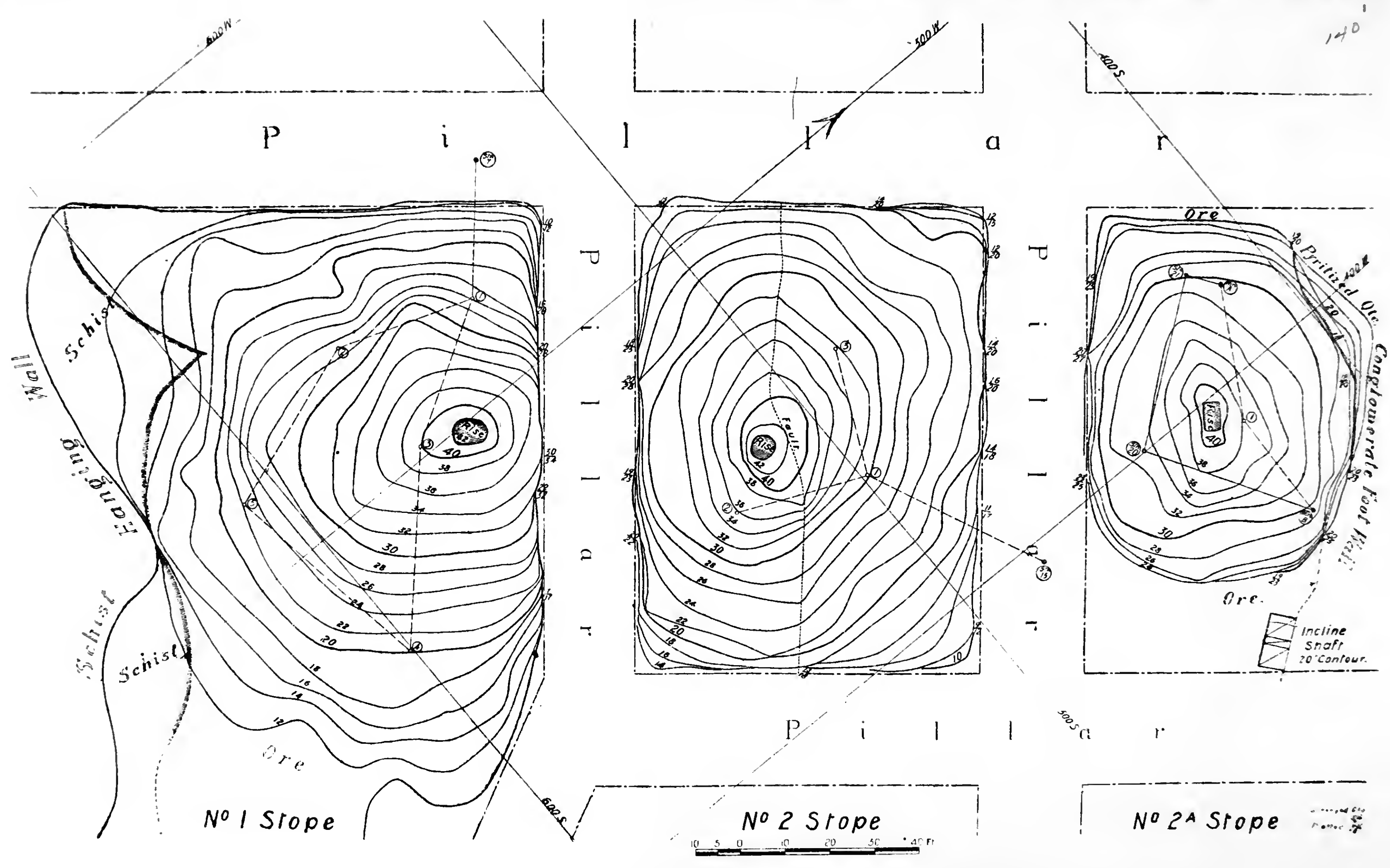
The mullock-pass system in the North Lyell mine is necessarily very extensive, both on account of the multiplicity of ore-bodies and the depth of workings. The mine is at present served by two distinct systems, each of which commences at the surface with one main trunk-pass, which branches on its downward continuation in a like manner to the roots of a tree, its ramifications ultimately extending over a very large horizontal area, and serving many stopes. As the stoping system requires the mullock-

passes to be located within the ore-bodies, it is necessary that a stope at any particular level must be filled from the pass traversing the stope at the level immediately above it, and, while this can be done without undue inconvenience over a limited number of levels, it could not be carried out indefinitely. Further, as the stopes at the upper levels become depleted the problem of maintaining the passes through large depths of filling presents itself. To overcome these difficulties, therefore, the main passes are kept within the hanging-wall as far as possible, the passes actually serving the stopes being branches off these, tapping them at suitable intervals.

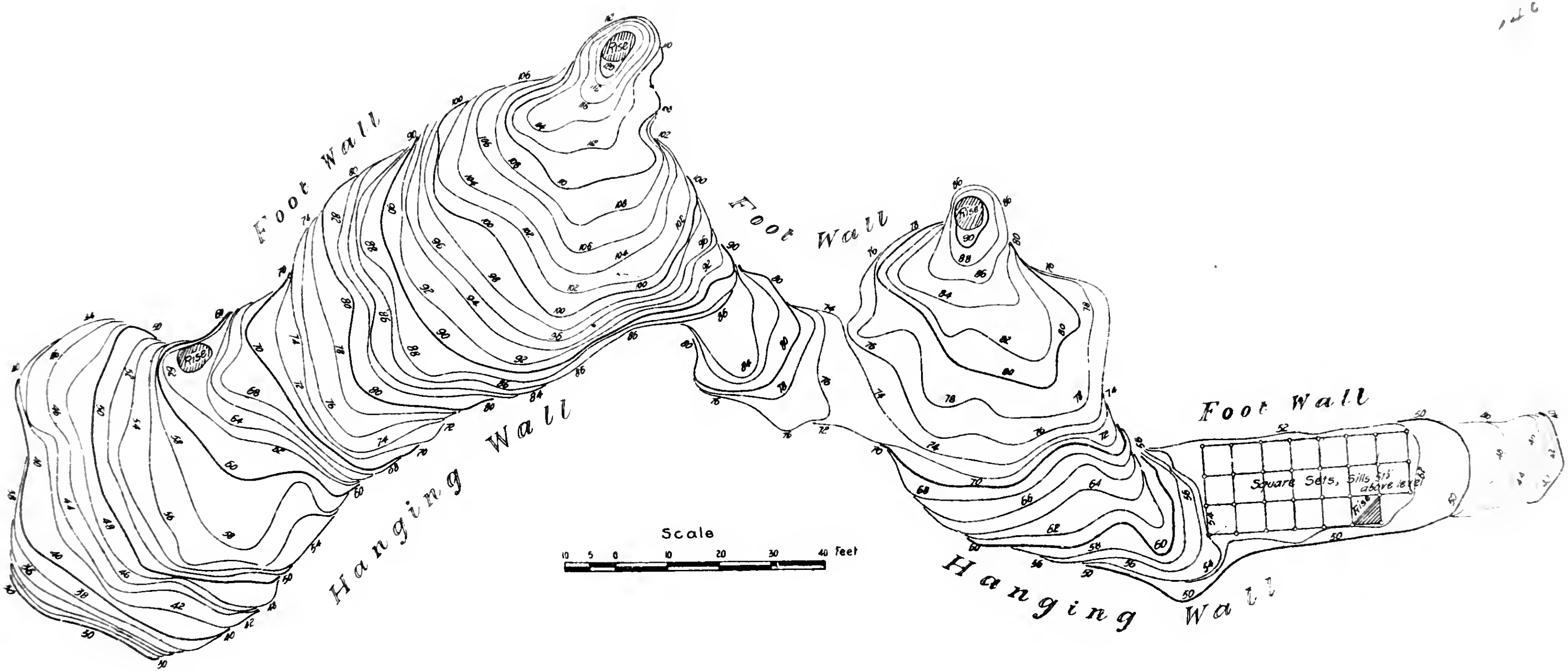
All passes are almost invariably on the underlay, varying from almost vertical to inclinations of 45°, which latter is the flattest angle at which the filling will travel freely.

The filling material is a mixture of the country schists and conglomerates, and is broken specially in the open-cut, from which all the ore has been extracted. The schist contains about 20 % of alumina, and the fines from it form, when wet, a stiff clay, which gives the excellent "setting" property to the whole filling mass. At the Mount Lyell mine the filling used is entirely schist.

All mine timber at both mines is used in the round as it comes direct from the bush, and is obtained along the Kelly Basin railway line and from the shores of Macquarie Harbour. The principal and best variety is eucalyptus, but a considerable amount of myrtle or beech, and some leatherwood, is also used. The cost delivered at the mines is about 5s. per 100 ft. super, based on the full solid contents of each log.



PLAN SHEWING CONTOURS OF STOPE ROOFS, NO. 7 LEVEL, MOUNT LYELL MINE.



PLAN SHEWING CONTOURS OF ROOF OF LARGE STOPE, 850-FT. LEVEL, NORTH LYELL MINE.

STOPE-SURVEY PRACTICE AT MOUNT LYELL.

By G. F. JAKINS AND L. J. COULTER.

THE method of stope surveying as practised at the Mount Lyell Company's mines has been evolved out of the exigencies created by special conditions. It presents some novel features, and has proved effective for all requirements. All stope-measurement work is brought under one complete system, to which office methods are standardized. Before proceeding to deal with the subject a brief description of the physical characteristics of the ore-bodies of each mine and the method of stoping will serve to show cause for departure from ordinary survey practice.

NORTH LYELL MINE.

The ore-bodies of this mine occur on or near the conglomerate-schist contact, and, being probably the result of replacements of both rocks, they rarely present any definite walls. It is usual for the payable material to merge into the respective country-rocks, and the limit of the occurrence is defined rather by assay value than by any distinct natural boundaries. The general configuration of these ore-bodies, therefore, is highly erratic both in horizontal and vertical section, so that, in order to obtain an accurate record of any occurrence, the measurement of the actual contour at short vertical intervals is essential.

Stoping is carried out on a modification of the "rill" system, perhaps more properly described as "arched back" or "dome" stoping. Mullock rises are placed in as convenient positions as the erratic configuration of each ore-body will allow, and around these rises the back is kept well arched, both longitudinally and transversely. The larger stopes are divided into arbitrary sections in which ore-breaking and mullocking are carried out in rotation, so that the whole of the excavation is seldom open for measuring at any one particular time. Stacking or square-setting on the

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mullock rill is employed where unfavourable ground is cut, and discontinued as the back becomes stronger. Square sets, closely followed by mullock, are used for picking up the level above, one or more floors being adopted, the number depending on local conditions. Fig. 1 typically illustrates the peripheral changes these ore-bodies exhibit between levels. The first outline shows the ground-floor at the 1000-ft. level, then follow contours at intervals of 10 ft. up to 60 ft. The last is the ground-floor plan at the 850-ft. level.

MOUNT LYELL MINE.

In this mine the ore-body is a lenticular mass of pyrites occurring at the junction of schist and conglomerate, the latter forming the foot-wall of the deposit. It is irregularly elliptical in contour, and pipe-like, or columnar, considered vertically. Taken generally, it is sharply defined against the enclosing rocks, and there is not the gradual shading into either schist or conglomerate that obtains with the North Lyell ore-bodies. Occasional inclusions of schist are, however, found.

Extraction has been carried out by open-cutting down to No. 5 level, the portion below this being worked by underground methods. Exploration work having established the dimensions of the body, it was possible to carry out stoping operations to a pre-designed scheme of pillar and stope mining. The pillars are 25 ft. in thickness, and the stopes are beaten out over floor areas varying up to 15,000 sq. ft.

Stoping is done exclusively on the arched-back method, each stope having a centrally situated rise connected to the mullock-pass system. The back is cut out dome-shaped from the commencement of overhead breaking. The picking-up of the level above thus begins around the rise, and the depleted area enlarges as the domed roof ascends until the whole of the ore is stoped to the pillars or walls. About 6 ft. of back is broken between successive mullockings, yielding a quantity of ore varying in the different stopes up to 15,000 tons.

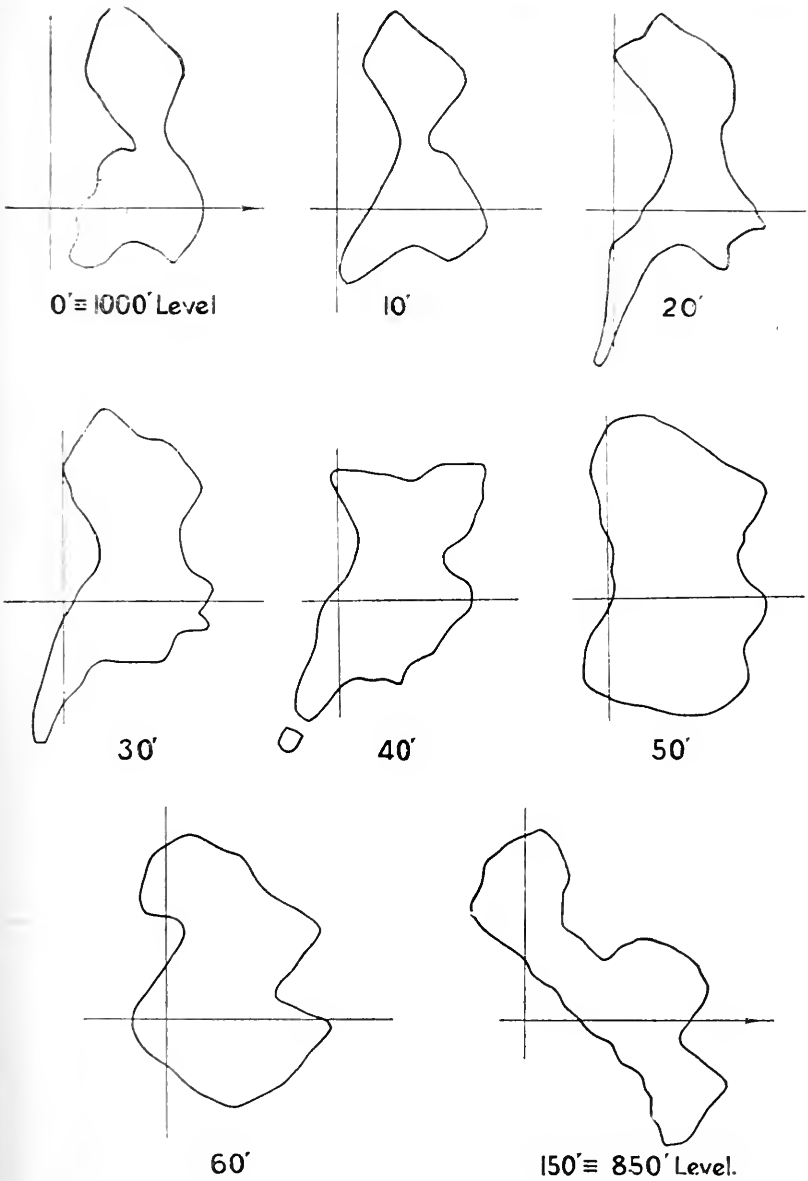


FIG. 1.
(Scale—80 ft. to 1 in.)

DESCRIPTION OF SURVEY PRACTICE.

At both the North Lyell and Mount Lyell mines the ground-floors are timbered wholly, or partly, with square sets. In the latter case gallery sets are stood to provide access to ore-passes and for through traffic. The periodical surveys of these ground-floors are performed by the ordinary methods observed in most mines.

The measurement of the overhead portion of these stopes, however, at the outset presented difficulties resultant on the scheme of stoping adopted. The highly irregular shape of many of the stopes, combined with the steep slope of the mullock rill, made the problem of effecting comprehensive surveys something beyond the practical scope of ordinary stope-survey methods. Under these circumstances the taking of parallel cross sections at equal distances was found to be too unwieldy in practice and insufficient for requirements.

The system inaugurated, which has been since successfully followed, is one of "contour surveying." By it field and office work are simple, and, for the extent of ground covered, rapidly performed, the suitability of the resulting "contour plans," both as records and for working guidance, being pronounced. It is of paramount importance that the shape of the domed arch should be carefully studied, especially in relation to outside factors that do not appear in an actual inspection of the stope. Exploratory work has frequently to be undertaken in stopes as they are carried up, and the accurate piecing together of all information gained is necessary to provide facilities for stoping where the ore-body develops some local eccentricity. It is of the utmost importance to have on record the precise shape and the geological features of the ore-body from the ground-floor upwards as a guide for laying out any subsequent work which may be necessary. In many of the larger stopes the picking-up stage is reached at the top of the dome before the lower portion has left the ground-floor, and owing to this large vertical range a considerable amount of fresh information becomes continuously available, from which necessary variations in the original scheme for working the lower portion can be made.

Stopes are measured before each mullocking, and also at half-yearly intervals, for the purpose of computing extractions and ore-reserves, and it is not common practice to measure stopes for ascertaining extracted quantities over shorter periods. Contractors are paid on truck tallies, and actual tonnages for costs purposes, &c., are allocated from four-weekly weighbridge receipts. The system of ore-transport also admits of intermediate quantity checks. It is only on rare occasions that it is found necessary to make measurements for progress payments.

The field-work performed in a contour survey consists in observing a number of points on the "walls" and "back" of the stope, so that their position and height above "floor-level" can be fixed on a plan and contour lines interpolated. Field measurements are effected by using both the horizontal and vertical circles of the theodolite in conjunction with the measuring tape, the surveyor requiring two assistants—a rod-man and a tape-reader.

Connection to the overhead portion of a stope is made from the level below by a direct horizontal and vertical traverse wherever practicable. This can generally be done in a stope where the lower portion has not left the "ground-floor." The instrument is set under the nearest permanent level-station, and a traverse run to the most convenient point for sighting up into the stope.

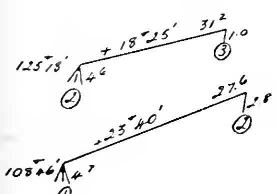


Fig. 2.

At this point the instrument is set, and its height above floor-level measured. The rod-man then drives the first peg on the mullock rill, and gives the surveyor a plumb-line sight for bearing. The rod is next held on the peg, a reading taken, and the vertical angle recorded. The tape is then stretched from the instrument to the observed point on the rod and the chainage booked. The notes are entered in diagrammatic form (Fig. 2).

Here 108° 46′ is the bearing from station (1) on the level to station (2) in the stope. The height of the instrument is 4.7 ft. above "floor-level," and 27.6 ft. is measured on an inclination of 23° 40′ to a point 2.8 ft. above station (2).

From station (2) the rod-man arranges for a third station in the stope, and this is traversed to from station (2) in a similar way, and so on until sufficient stations are placed in the stope from which to make the required observations. In these traverses angles are booked to the nearest minute, and chainages and heights to the nearest tenth of a foot. Angles of elevation are marked (+) and depression (−). The first set of observations is taken from station (2), and in the following way :—The instrument is set up for both limbs and placed in meridian by setting the back bearing, sighting station (1), and plunging. The rod-man clips the ring of the tape and a lighted candle to the end of the rod and holds it against any selected point on the wall.* The tape-reader holds the box at the instrument and calls the measurement, paying out or winding in as is required. The bearing and vertical angle are read, and the rod is then moved toward the instrument in the same vertical plane to another point, which is observed in a similar way. After this, additional points are taken, as considered necessary, still on the same bearing.

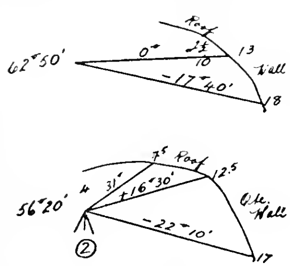


FIG. 3.

The notes are booked in sketch form (Fig. 3).

The rod-man then moves to another point along the wall, and further observations are taken on a new bearing, and so on until a radiating set of sections has been completed around station (2). The instrument is then moved to station (3), where the procedure is repeated, and so on until the survey is finished.

The angles in these observations are booked to the nearest ten minutes and distances to the nearest half-foot, it being needless refinement to attempt greater accuracy on a rough surface such

* A convenient type of rod for this work consists of two pieces of 1-in. "tamping stick," each 5 ft. 6 in. long. These can be fitted together to make an 11-ft. rod by slipping the end of one piece into a ferrule fitted on the other. The rod is painted in feet with alternate black and white rings. The end of the rod is fitted with a clip to hold the candle, and the ring of the tape is also fastened to it. The tape used is an ordinary 66-ft. metallic box-band.

as a stope presents. The angles are read directly from the plates, without employing reading-glass or vernier scale. In this way shots are very rapidly taken, and a stope requiring, say, three observing stations can be measured with ground-floor connection easily in $1\frac{1}{2}$ hours. A surveyor generally reckons 15 to 20 minutes for each station, at which he will take from 40 to 60 points after setting up his instrument.

Judgment has to be shown in selecting stations which will give a good command over the area to be observed. Stations involving measurements to points more than 40 ft. distant are generally inadvisable, and the survey is better effected by inserting an extra one. The surveyor should avoid picking up small local crevices and bumps, which are unimportant in themselves, and needlessly distort the contour lines. On an average, adjoining sections should intercept the walls about 6 ft. apart, and each section should not be followed back too close to the instrument, else a surfeit of information will be collected round the station. Only sufficient points should be taken to ensure a good general survey of the roof.

In a stope where the whole working has left the ground-floor a satisfactory connection can sometimes be obtained by sighting up an inclined ladderway, or use may be made of an underlay winze from the level above. Occasionally a line may be established up in the stope itself along a timbered gangway, etc., which may be left open enough to afford connection for several successive surveys.

Where such means are not available, and the stope is a large one, two ladderways are selected, one at each end of the stope. If possible, a wire is hung in each from the top, and connected to the permanent traverse on the ground-floor. When the ladderways are too crooked to admit of this being done, a combination of plumbing and traversing has to be effected, generally with the aid of wall plate and diagonal eye-piece, care being taken during such work to make permanent any points likely to be of service in subsequent surveys.

The two points thus established in the stope above are included in the stope-traverse, which is conducted on an observed magnetic

meridian, and afterwards corrected by calculation with the bearing between the wires on the level below. Such a case is shown in Plate I. In measuring this stope two ladderways were used, and wires A and B were suspended and observed from stations (12) and (16) on the ground-floor. The instrument was then taken into the stope, and set on station (1), where a magnetic needle observation was taken for a meridian on which all bearings read in the stope were temporarily based.

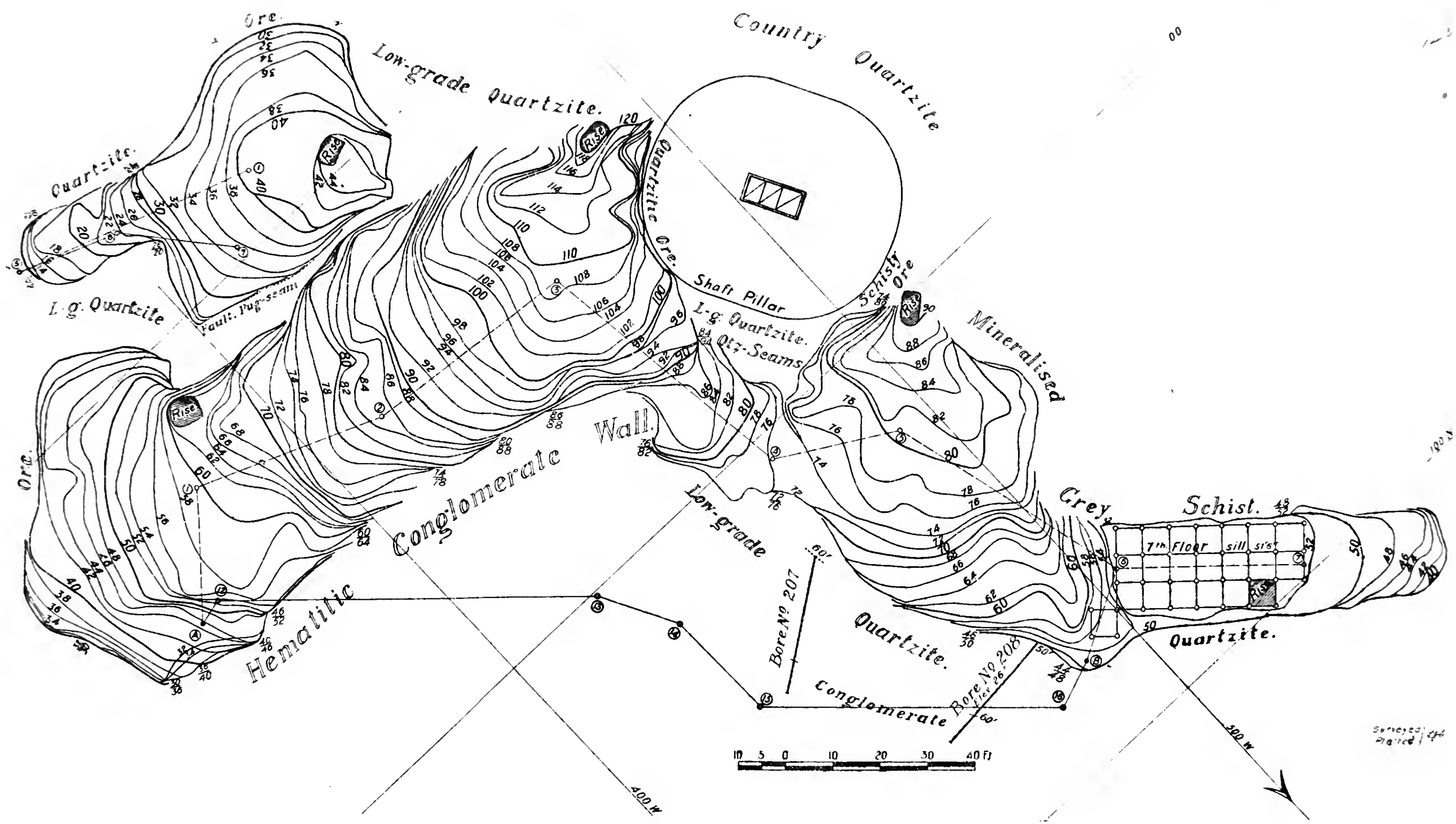
On the completion of the survey the bearing on the level between (A) and (B) was calculated via stations (12) to (16), information from the office co-ordinate book being used to facilitate the computation. The bearing between (A) and (B) was then calculated via stations (1) to (6) in terms of the temporary stope meridian, and the correction to all stope-bearings entered in the notes, so that the adjustment could be made in plotting.

In this work the length AB in both traverses is also computed, and gives a useful check on the accuracy of the stope-traverse. Another good check is obtained on the vertical heights as measured up one ladderway, through the stope by medium of instrument and tape, and down the second ladderway. Any difference of floor-level at A and B being known, the vertical co-ordinates of the stope-traverse can be then balanced. As these calculations can be effected in a few minutes with a slide-rule, the surveyor generally prefers to so balance his work before he leaves the stope.

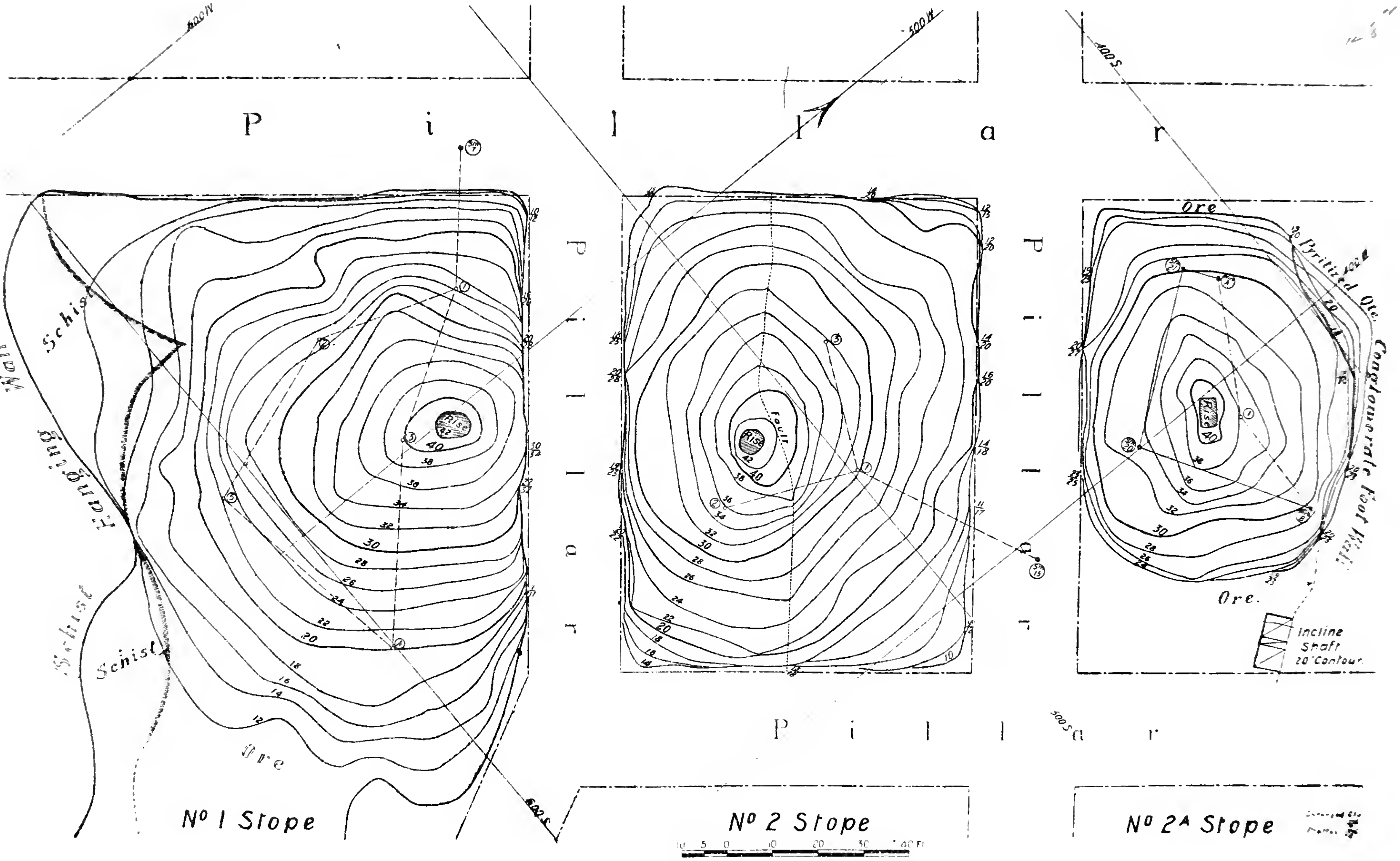
Wire connections may be similarly made through vertical winzes from the level above.

Where a small portion only of a stope is open and requires measurement, a single-wire connection is made from the floor-level. or level above, to the stope. A magnetic observation is taken in the stope for meridian, and adopted for the survey. If ordinary precautions are taken against local attraction, this meridian can be relied upon within 20 minutes, and is sufficiently close for the small area to be surveyed.

Plate II. shows contour plans of stopes across the Mount Lyell ore-body. In Nos. 1 and 2 stopes direct traverses were made



CONTOUR PLAN OF STOPE, NORTH LYELL MINE.



PLAN SHEWING CONTOURS OF STOPE ROOFS, NO. 7 LEVEL, MOUNT LYELL MINE.

into the stopes from cross-cuts through the pillars on the ground-floor. The arrangement of the observing stations is also shown. In the case of No. 2A stope a two-wire connection was made, a single observing station being sufficient for the measurement. In No. 1 stope the schist hanging-wall exposed is marked, while in No. 2A stope a flat portion of the footwall is shown between the 18-ft. and 22-ft. contours.

Field notes are kept on the "loose-leaf" system, the notes being filed in the office in special books kept for each stope. All calculations of connections are made and tabled in the field book. Before going underground to make a survey the notes of any previous connections likely to be of service are transferred from the office file to the field book. One connection can sometimes be used for several surveys, or the connection of an old survey may be slightly modified or extended to serve a subsequent one.

Contour plans are drafted on tracing linen to a 10-ft. scale, and contour lines are shown 2 ft. apart vertically. The office work of a contour survey commences with the reduction of the field measurements into horizontal and vertical components, and this reduction may be effected either graphically or by calculation. The former method has been superseded by the latter, with marked advantages. In view, however, of the fact that the method of calculation involves the use of the slide-rule to be practicable, it may be interesting to describe the "graphic" method, which successfully dealt with these reductions before the slide-rule method was adopted.

The graphic method consists in first preparing a separate graph plan, on 10-ft. scale, on which all measurements are plotted in one vertical plane. The co-ordinates are then scaled and transferred to the contour plan on their several respective bearings.

The graph plan is first prepared by ruling it with parallel horizontal lines to correspond with the 2-ft. contour intervals. Then, to instance the method of procedure, if we take the notes given in Figs. 2 and 3, station (1) is first placed on the graph, with the height of the instrument 4.7 ft. above floor-level. The

line from (1) to (2) is then set off on an elevation of 23° 40′ and length 27.6 ft. with protractor and scale, the position of (2) being obtained by measuring down 2.8 ft. from the end of this line. The height of the instrument over (2) is then marked, and the line (2) to (3) set out in a similar way. The sections taken from (2) (Fig. 2) are then plotted.

To save needless work, and to avoid complicating the graph with a multiplicity of lines, it is advisable to omit the collimation lines, plotting only the actual section of the ground.* Each section is marked with its bearing as a guide when transferring to the contour plan. The graph of notes contained in Figs. 2 and 3 is shown in Fig. 4.

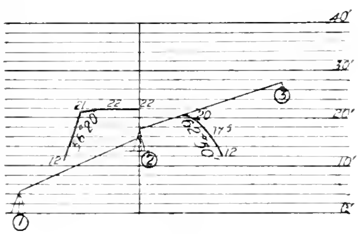


FIG. 4.

The contour plan is first prepared by ruling on the standard co-ordinate squares of the mine. The level and stope-traverses are then plotted, the horizontal components of slope measurements being transferred from the graph. The protractor is then set over each observing station, and, by the aid of the graph, the information obtained by each section is transferred on its correct bearing. In this way each point on the walls, with its height above floor-level, is entered; and the intersections of each section, with the contour lines of graph, are located on the contour plan. When this work has been completed for all stations the contour lines are drawn by connecting the points of equal altitude, and the plan completed.

The second method of reducing the survey measurements to horizontal and vertical co-ordinates—*i.e.*, by calculating the actual heights and distances—is to be preferred, provided the results can be conveniently attained and entered in the field notes. The

* The following has been found a quick and convenient method of plotting these sections:—An 18-in. cardboard protractor is centred and adjusted over the observing point, through which point a small pin is then driven. The scale, which has a small notch cut on the zero mark to engage the pin, can then be swung round the pin on the protractor, and the different points marked directly on the graph.

time involved in plotting the graph is saved, and the draughtsman plots direct from the field notes all information gained by the survey. In addition to the convenience of this, he is better able to reconcile the different results gained in the survey, and a more reliable plan is likely to result. A means has, however, to be selected so that these computations can be quickly and easily performed. Mathematical tables must be set aside as too cumbersome for such a purpose, but a "diagram" may be brought within range of practicability in spite of its complicated appearance. Such a diagram may be prepared by marking angles from 0° to 90° by lines radiating from an origin and setting out slope distances by arcs centred at it. The vertical and horizontal components of any radius on any vertical angle may be then obtained by reading the corresponding ordinate and abscissa, the diagram being ruled with horizontal and vertical lines to facilitate the operation.

The most efficient means, however, is the slide-rule. In this work reductions are easily made with the rule and entered in the notes at the rate of 250 observed points or 500 calculations per hour. The cursor is unnecessary, and may be removed from the rule. In actual practice it is found that mistakes are singularly rare, and are almost invariably detected in plotting, so that, while it was formerly the custom for the draughtsman to check the surveyor's reductions, the plot is now regarded as sufficient safeguard against error. It must be pointed out that the measured lines are short, and the limits of accuracy observed in the field are 10 minutes and .5 ft., so that any attempt at greater accuracy than .5 ft. in the reduction of these measurements is unnecessary.

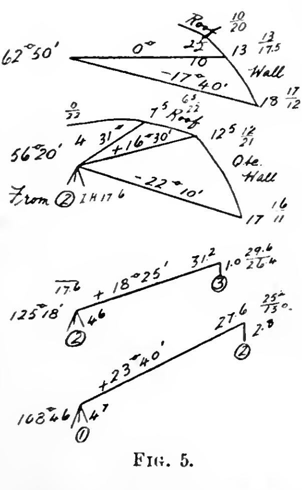


FIG. 5.

The reductions for the notes contained in Figs. 2 and 3 are entered in the field notes as shown in Fig. 5.

In each pair of reductions the upper figures relate to the calculated horizontal components, and the lower to the calculated vertical heights above floor-level.

The plotting of the contour plan proceeds as follows:—The stope-traverse is drawn in the ordinary way, except that horizontal components of slope-measurements are found in the notes. The protractor is then set over each station in turn, and the observed points with their vertical heights above floor-level marked. The area thus becomes covered with a number of level-points, from which the contour lines are interpolated.

The preparation of a contour plan thus made takes from one to five hours, according to the area of stope measured, and the detail required apart from the contour lines. The total time saved over the graphic method is one-third. This, of course, includes all reductions made by the surveyor.

In preparing a contour plan, the use of coloured inks based on a conventional system has distinct advantages in presenting a plan clear in detail. For instance, floor-level traverses are shown red, stope traverses black, contour lines sienna, underfoot contours blue, and so on.

Before the finished contour plan is filed it is used for the preparation of the following plans and sections:—

General Section.—A representative longitudinal and transverse section is struck and added to a plan showing a series of such sections arranged in composite chronological order. This is useful in presenting a general idea of the doming and the progress of exploitation of the stope.

Foreman's Tracing.—A plan is prepared, for the underground foreman, of any stope in which one or more fixed boundaries, such as pillars, have to be observed. The foreman's working guide to a pillar generally consists of two or more ore-passes which are reared against it. The positions of these passes are located each survey, and entered on the tracing, which also shows the position of the pillar, so that any tendency towards encroachment, or want of alignment, may be ascertained and corrected.

Ten-foot Contour Plans.—The contours at 10-ft. intervals above floor-level are reduced by pantograph to 30-ft. scale, and plotted on a special set of plans kept for superimposing on the working-level-plans, for general information, and also for model-building purposes, &c. Each contour, which bears the date of survey, is dotted until it reaches the wall of the ore-body, along which it continues in a full line. Fig. 6 shows a 40-ft. contour of a stope which has partly reached the walls.

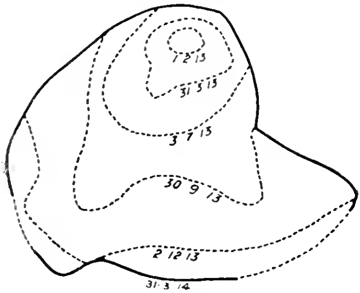


FIG. 6.
(Scale—60 ft. to 1 in.)

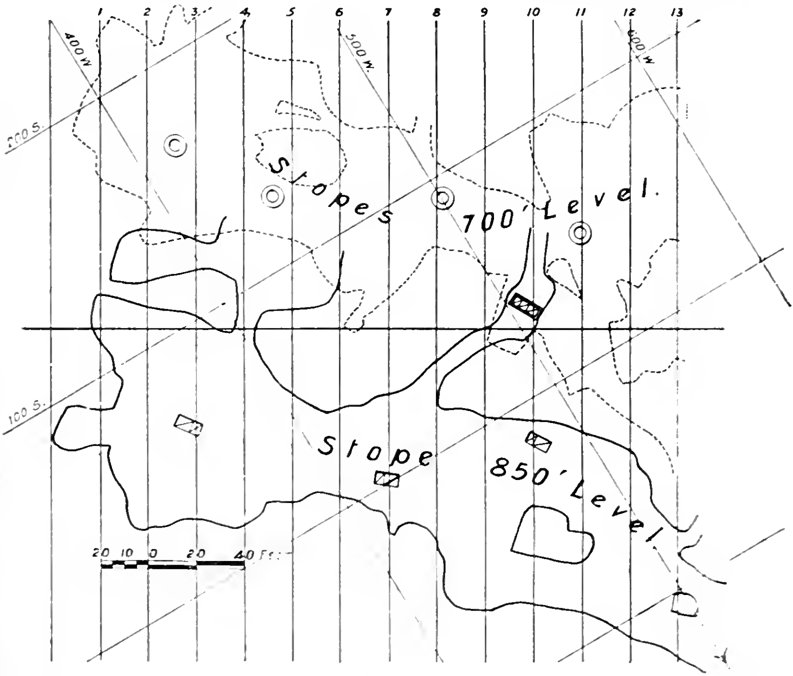


FIG. 7.
(Scale—80 ft. to 1 in.)

Ore-Reserve Sections.—Ore reserves are computed from a series of sections along parallel lines through the ore-body between two

levels. The direction of these lines is made as transverse to the body as its known irregular shape will allow. The lay-out of a typical set of sections is shown in Fig. 7, which deals with a portion of an ore-body between the 850-ft. and 700-ft. levels. The lines are 20 ft. apart horizontally, and numbered in rotation. The respective sections are compiled from the half-yearly contour-surveys, and form a special set of section plans, drawn to a 30-ft. scale, used for quantity purposes. On each section the form of the portion of the ore-body remaining in situ between the levels is then defined in accordance with all available information concerning it, and the area measured by planimeter. When this operation has been performed for all sections the volume in reserve is calculated by the prismoidal formula, and the quantity extracted during the expired half-year is similarly computed. Sections on lines 5, 6, and 7 of Fig. 7 are shown in Fig. 8, and fully illustrate the method. The erratic changes in section of this particular ore-body as stoping progressed are also shown.

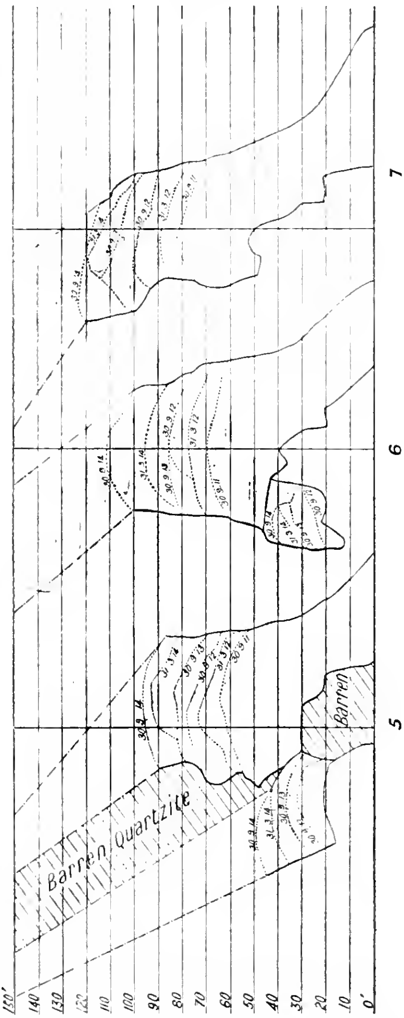


Fig. 8.
(Scale—80 ft. to 1 in.)

The finished contour plan is finally filed in the booklet of plans of that particular stope. It is inserted in correct co-ordinated position over the plan compiled from the previous survey, with which it can be closely compared through the transparent linen. Any alterations in form of the ore-body that may be developing can thus be studied, and information obtained for laying out any exploratory work deemed necessary, or for varying the existing scheme of stoping.

The special fitness of such a working-plan continually presents itself as providing means for considering stopes in a concrete manner with much greater convenience than is possible with a series of detached contours or sections.

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME, MOUNT LYELL, TAS.

By GEO. W. WRIGHT.

THE exceptional rainfall characteristic of the West Coast of Tasmania, in conjunction with the rugged topography of the district, has always suggested the utilization of water-power, and from the earliest days of the Mount Lyell Company the eventual installation of a hydro-electric power scheme, making use of some near-lying lake, has constantly been kept in view.

At the period of the inception of the company, however, the time was not ripe for such an undertaking, and it is only of late that the various important considerations involved have sanctioned incurring the heavy expenditure necessary, with the positive assurance of permanently reaping the benefits expected.

The West Coast of Tasmania contains many lakes of various sizes, situated at high levels, and, although the catchments of these lakes are not very extensive, still, the excessive rainfall is so constant over long periods that its regularity may be relied on as offsetting the limited catchments. The explanation of the abnormal rainfall is simply the circumstance that the West Coast Range forms the first high land barrier which is met by the moisture-laden westerly and north-westerly winds blowing off the Southern Ocean. By the time the clouds strike the mountains they ride very low, consequently the elevation of the range, the peaks of which do not exceed 3000 to 4000 ft. above the sea, is sufficient to cause a copious precipitation.

Lake Margaret, which is situated five miles north of the reduction works, was first recognized as a possible source of water-power by Mr. Hugh Chrisp, a surveyor, who visited it about 1893–4. Early in 1895, while the preliminary work on the erection of the first furnaces of the Mount Lyell Co.'s reduction

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works was in progress, the company sent Mr. C. Wadsworth James, C.E., to make a survey and report on the possibility of the near-lying idea to utilize the lake and the great fall at its mouth. This was done in March to June, 1895, but the conclusions arrived at were not considered sufficient to warrant any action being taken. The locality was difficult of access, the cost of construction prohibitive, cheap firewood was abundant for steam generation, and the proper transmission of electric power over long distances was still a problem.

In the summer of 1901 the lake was visited by prominent members of the staff, who spent several days in the vicinity, and whose observations then have since afforded the nucleus of the present scheme.

In the following year (1902) a rain-gauge was erected on an island in the lake, and periodical readings were taken up to January, 1905. In the interval, and afterwards, the scheme received further occasional attention, and the general outlines of the undertaking gradually took shape, but, as the economic inducement, based on all the considerations which came into play in a mining enterprise, was not considered sufficiently certain to warrant the large outlay, the whole matter was again laid aside. Finally, however, firewood had become scarce, coal was dear, electrical machinery and transmission had been much perfected, and other circumstances contributed, so that, early in 1911, the scheme was again revived, and in March of that year Mr. A. G. M. Michell, M.C.E., of Melbourne, was invited to specially investigate the subject of Lake Margaret and the power derivable therefrom, as proposed by the engineering staff. Assisted by the latter, Mr. Michell made a careful investigation of the project formulated by the company's officers, and was sufficiently impressed by the possibilities to report favourably, so that, in due course, it was decided to undertake the work. Owing to the strike of the company's employés in September, 1911, the initiation of operations was delayed until June, 1912, when a start was made with the necessary preliminaries for access to the lake.

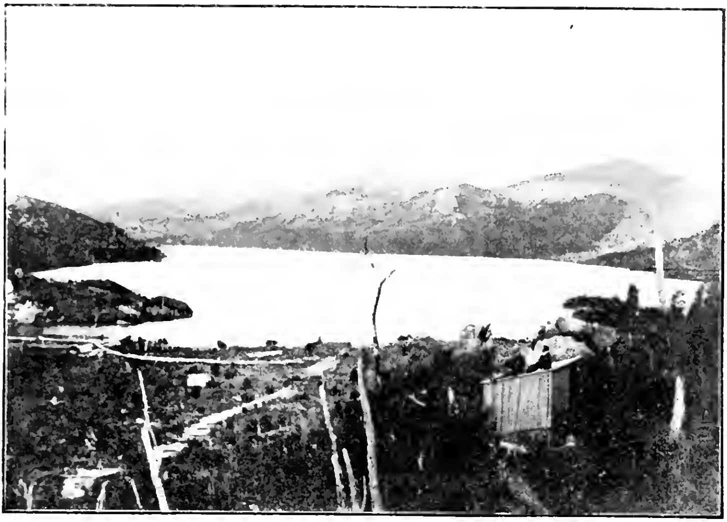


FIG. 1.
Lake Margaret.

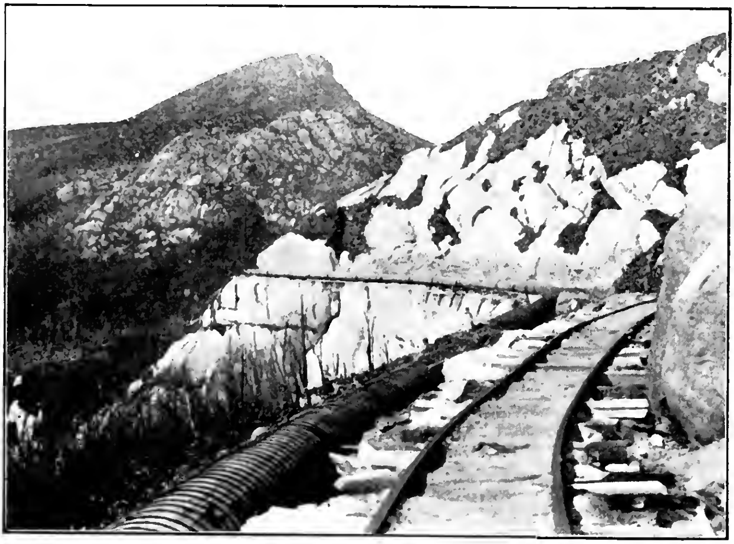


FIG. 2.
Wood Pipe-Line, Cliff Section.

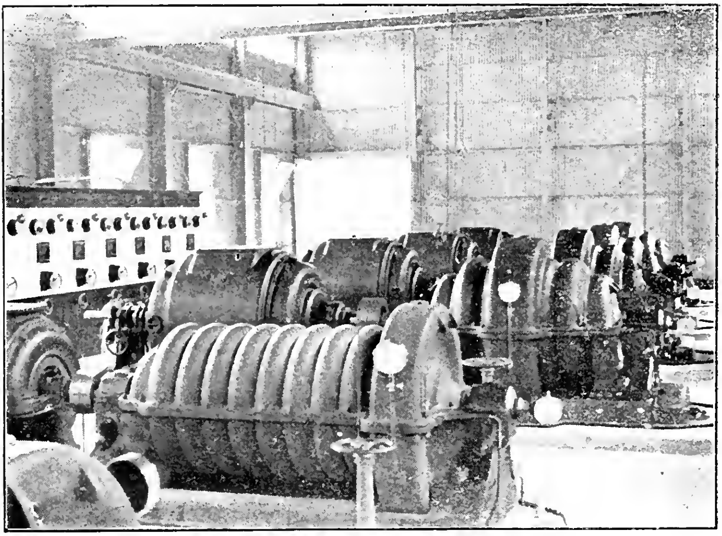


FIG. 1.
Blower House Interior.

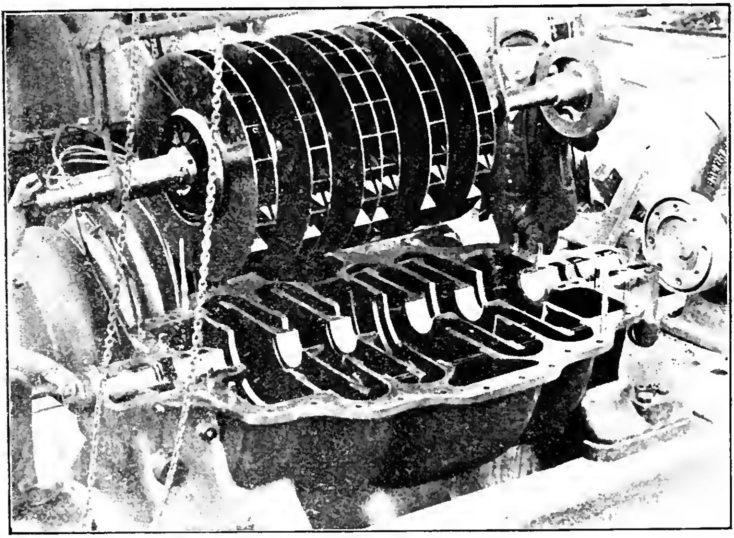


FIG. 2.
Detail of Turbo-Blower.

GENERAL FEATURES.

A water-power scheme of the character of the Lake Margaret installation must possess certain features to make it commercially successful, amongst which are—

1. An adequate catchment area.
2. Suitable site for a storage reservoir.
3. A regular and heavy rainfall.
4. Short length of pipe-line.
5. Suitable power-station site.
6. Proper means for transport of machinery, &c., at a reasonable cost.

The Lake Margaret scheme possesses all these attributes in good measure.

CATCHMENT AREA, RAINFALL, &c.

The catchment area from which the water is obtained forms a roughly circular basin, containing $7\frac{1}{4}$ sq. miles. The edges of this basin are formed by the serrated conglomerate ridges and spurs of Mounts Sedgwick, Tyndall, and Geikie (Plate V.), with an average altitude of 3000 ft., with the exception of the short and narrow U-shaped gap forming the lake outlet. The elevation of the latter is 2150 ft.

The lake, which forms a natural reservoir of about 303 acres in area when full, is thus located in what is termed a "hanging valley," and the descent, from its mouth to the lower ground fronting the range, is sudden and precipitous, thus affording the necessary head in a short horizontal distance.

The lake trough is doubtless the result of glacial action, of which evidences abound in the locality, and its greatest depth is about 150 ft., but remote from the outlet. The latter has shallowed up, owing to the presence of a resistant bar of the conglomerate formation, extending well above the lake-level, which closes the greater part of the gap at the very edge of the descent, and now practically forms the dam of the storage scheme. The distance from this bar back to the edge of the lake-trough

is only about 12 chains, and this place was occupied by a marshy flat with sand bottom. Out of the flat issued a twofold stream, flowing through depressions at each end of the natural dam, and uniting in a series of picturesque cascades and waterfalls on the precipitous outer declivity, to form the Yolande River in the small plain below, some 500 ft. down. This river flows into the Henty River, and its descent is so rapid that, at a distance of $1\frac{1}{2}$ miles from the lake, the fall is 1100 ft.

It will be seen from this brief description that the configuration of the lake mouth was exceptionally favourable for a cheap storage scheme, and the usual heavy expenditure for a proper dam could be entirely avoided.

The rainfall at Lake Margaret averages about 150 in. per annum, and is fairly evenly distributed over the four seasons, with no marked wet or dry periods. In the winter time there is some snow, but the lake never freezes over. The evenness of the precipitation is a most valuable factor in the general scheme, for it obviates the necessity of a large storage, and has enabled the installation of a plant supplying 5000 h.p. on a reserve of only 16 ft. of water over the lake area.

A very useful diagram (Plate VI.), known as "Rippl's" method, was used to determine the necessary height of dam required to store enough water to ensure getting the required quantity at all times. On the base line time is marked off in convenient periods (in this case 7 days). Starting at the left-hand end of the base at each of these periods the cumulative run-off is plotted vertically to scale, giving a curve, as shown. A rate line is also drawn, which can be transferred on to any part of the run-off curve. If the rate line corresponds to the run-off curve there is neither more nor less water than is required; but, should it lie above the run-off curve, the vertical distance between the two represents the amount of draw, in cubic feet, on the storage. This can be translated into terms of depth by dividing this amount by the area of the storage at this point. By this method the driest periods, requiring the greatest storage, can be seen at a glance, and the requisite height of dam determined.

PIPE-LINES AND POWER-HOUSE SITE.

The length of conduit from lake to power-house is short, the total being under 2 miles, in which distance a total static head of 1100 ft. is obtained. This compares more than favourably with any of the electric schemes coming under the writer's notice. It was a fortunate circumstance, as the preparation of pipe-line beds in such rough, broken country was an expensive proceeding.

The natural disposition of the country provided an excellent site for the power-house without the excessive cost of excavating out of the high bank of the river, which would have been necessary had any other site been selected. The place chosen is situated in the valley of the Yolande River, below the falls, at the foot of a steep spur running off the slope of Mount Sedgwick, which forms the left-hand bank of the river. This spur has a smooth descending profile, which made the erection of the pressure pipes a simple and easy matter. The starting point at the top of the spur, however, is some distance away from the lake mouth, and it became a question how to lead the water from the latter to the pressure pipes. Whilst this is ordinarily done by open race, preference was given, in this instance, to a pipe system, conveying the maximum quantity of water to be used—*i.e.*, 45 cub. ft. per second—for the present, and a wooden conduit pipe was selected on account of cheapness and greater ease of installation.

The position of the power-house is such that the steel pipe-line is almost a straight line (Plate VIII.) until it reaches the distribution pipes alongside the power-house. This is a very great advantage, as the elimination of changes in grade and direction minimizes the amount of concrete required for anchoring the pipes at these points.

TRANSPORT FACILITIES.

Whilst the cost of providing transport facilities was high, it was not nearly so great as it would have been years ago, because the result of pushing out the firewood-getting in the direction of the power-house left only about 4 miles of tramway to be made. This circumstance contributed in no small degree towards the

quick installation of the scheme. To render the various localities accessible to the main railway system a branch of the existing 2-ft. gauge steel tramway on Howard's Plains (which had hitherto been used for the purpose of the firewood supply) was extended to the site of the proposed power-house—a distance of $4\frac{1}{2}$ miles—from a convenient terminal point, $2\frac{1}{2}$ miles from the gravity incline leading down from the elevated plateau of the plains into the reduction-works valley, 420 ft. below. This incline connects this high-level tram system direct with the company's general tram and railway system, terminating at Regatta Point, on Macquarie Harbour.

The route of the extension was through very rough country, including much rock cutting, heavy filling, and many bridges. The work was carried out under exceedingly unfavourable weather conditions, but the men engaged stuck to their task manfully. The construction of the last 2 miles, however, promised to be a very lengthy undertaking, owing to the exceptionally heavy work. A wooden bush tramway was therefore temporarily substituted for the steel tram over this portion, on a shorter route, with steeper grades. The final line to the power-station site was completed by the end of the year 1912. Incidentally, the wooden tram, as well as the steel tram, were used for the transport of practically the whole of the firewood requirements of the reduction works during the construction period of the power scheme, and the country passed through also contains a good supply of King William pine, otherwise now scarce, which may be of use for wooden conduit pipes, &c., in the future.

To complete the transport system, an inclined steel-tram haulage-line was constructed from the site of the power station, up the above-mentioned spur, to about the level of the lake mouth, for the purpose of taking up the pressure pipes and the large quantity of timber, cement, and other materials used on the high-level works. This was served by a 40 h.p. electric winder on the top of the haulage. The latter is as steep as 1 in $1\frac{1}{2}$ in places, and has a vertical height of 1047 ft. The final connection, which finished the line of communication between the

lake and the reduction works (and incidentally Regatta Point), was a wooden tram laid from the site of the upper end of the pressure pipes to the lake, nearly on a level, the fall being only 50 ft. in a distance of 1½ miles. This tram skirts the sinuous outline of the hill from the spur to the lake mouth, which contour is also exactly followed by the wooden conduit pipe. The work included the excavation of a bench around a perpendicular cliff, cut 12 ft. wide out of hardest conglomerate rock, for a length of 500 ft., to take both the tram and the wood pipe. This high-level route was practically finished by January, 1914, and, once all points were connected up, construction work was proceeded with in many places simultaneously.

The first necessity at the power site was a preliminary power plant for use in construction work, and for this purpose a temporary hydro-electric plant was installed in the vicinity of the present power-house, driven by water from the Yolande River, drawn from up stream, some distance away, and conducted by a wooden flume, laid along the precipitous sides of the river, into a large tank about 240 ft. above the temporary power-plant site. From this tank a pressure pipe, 16 in. diam. at the top to 12 in. at the bottom, conveyed the water to a turbine, driving a three-phase generator of 60 kw. capacity. To all intents and purposes this small plant was a miniature duplicate of the larger one, and has served its purpose admirably. It was chiefly used to operate the electric winder on the haulage line, and also to supply power for driving the wood-working machinery used in connection with the construction of the wooden conduit pipes. In addition, it served the electrically-operated percussive drill used on the heavy rock work along the route of the wooden pipe-line, and supplied the necessary light for night work in the vicinity of the lake. Owing to delays in the arrival of necessary parts, this power plant was not finished until June, 1913. The pipe-line for this particular service has been of great assistance as a water supply during construction work and as a very efficient fire service. It has since been used for the reticulation of the houses around the power-station. The plant is likewise an independent source of

power in case the main plant should be shut down, when it would probably be required for service in operating the electric winder, and for light.

CONSTRUCTION OF DAM AND RESERVOIR.

Reverting to the question of the storage of water in the lake, two methods were available. The first was to build a dam of the necessary height on the top of the natural rock bar above mentioned, and the second was to cut through this bar to the requisite depth required and to use the natural storage of the lake. For reasons of expedition and cost, and also with an eye to future enlargements of storage, the second course was adopted. A cutting was put through the rock bar at a low point, following the natural channel already worn in the rock by the principal stream, close to the precipitous walls of Mount Sedgwick. Fortunately, the material behind the rock bar—*i.e.*, towards the trough of the lake—was only alluvial and detrital stuff, through which a channel could easily be formed by ordinary sluicing operations. The distance from the cut, back to the edge of the actual lake, is only 800 ft. The clearing of this alluvial flat was necessary for the purpose of being able to build in a concrete retention wall or dam, containing discharge pipes, &c. The second overflow of the lake, which was all in sandy formation and could easily be deepened, facilitated work in the cutting, and, as soon as the proper depth was reached, the water was allowed to run out of the lake through the cut and the latter was afterwards closed with a secure concrete wall, in which the pipes and screens are permanently set. The excavation of the channel through the rock bar was exceedingly laborious, as already mentioned, and it, as well as the building of the wall, were greatly interfered with by periodical floodings during heavy rains. It was not until 8th August, 1914, that the last of the rock was removed, and sluicing operations commenced. Although the difficulties thus encountered in the cutting were vexatious at the time, yet they afforded good evidence of the soundness and reliability of the rock bar as the chief element in the closure of the

lake mouth. The erection of the concrete wall across the cut was commenced on 9th October, 1914, and finally fully completed on 23rd December, 1914, after the power was already in use. The net depth of storage thus secured between the by-wash level and the outlet pipe is 16 ft. This depth, compounded with the area of the lake, represents the present storage capacity of 198,000,000 cub. ft., but is, of course, not final, inasmuch as the present lake level can easily be raised.

The total amount of storage required to carry over the driest period on record is only 11 ft., with the present maximum discharge of 48.33 cub. ft. per second.

The dam is thus of very small dimensions, only 835 cub. yds. of concrete being used in its construction, and is scarcely a "storage dam" in the usual sense of the term. It is built thick enough at the base to permit of being increased in height to the full extent which may be necessary at any time in the future. The adjacent natural rock bar has its top now 26 ft. above the lake, and can, of course, also be heightened. The stone used for the dam was the local conglomerate rock, and the sand was obtained from the alluvial flat in front of the rock bar mentioned above. The broken stone, sand, and cement constituted 62 % of the total mass of concrete, the remaining 38 % being made up with displacers of conglomerate boulders and large pieces of broken conglomerate rock. The quantity of cement used was 0.82 of a cask per cubic yard of concrete. When the water was impounded not a leak of any kind from the dam was observed. Part of the concrete was mixed by hand and part with a concrete mixer.

Two massive cast-iron outlet pipes, of 4 ft. internal diameter, are set in the concrete wall, and terminate, on the up-stream side of same, in two screen compartments (Plate VIII.) each fitted with double vertical screens of woven wire mesh, sliding in cast-iron guides set in the concrete, each screen being raised separately, when required, for cleansing. Between the lake and the screens flat sluice gates are provided, to allow of access to the screen chambers when these are closed. These sluice gates are made of cast iron

and slide in cast-iron guides built into the concrete. Both the guides and face of the sluice gate were machined, in order to ensure a water-tight joint. On the down-stream side of each pipe, just outside the dam, there is a 4-ft. sluice valve, which can be operated either by hand direct or by means of electrical motor controlled from the power station switchboard, 2 miles distant. At present only one of these valves is required, the second being there for future extension—*i.e.*, for an additional wooden-pipe conduit, etc. Excepting the sluice valves themselves, the rest of the paraphernalia, such as the pipes, screens, sluice gates, etc., were made in the company's shops at Queenstown.

The by-wash outlet of the lake, on the other end of the rock bar and close against the side of Mount Geikie, was closed by a simple earthworks dam of proper height, provided with a spacious overflow notch, or weir.

PIPE-LINES.

Wood Pipe-Line Construction.—The pipe-track formation passed through some very broken country and entailed a considerable amount of work in removing conglomerate boulders and cutting through rock in order to procure a uniform grade throughout. It may be mentioned that blasting gelatine was used for breaking up the boulders. The method of using was to place a plug of the explosive on top of the boulder and cover the same with an air-tight covering of clay, and then exploding in the usual way. It proved to be of immense service in quickly disposing of boulders. The work of rock-cutting generally was much expedited by the use of an electro-pneumatic percussive drill, driven from the temporary power-plant mentioned before, a cable line being taken from the temporary plant right to the lake. This also provided light for night work, which was considerable.

Bearers were placed at 8-ft. centres on the formation, or on elevated trestles where they were required, and the wooden conduit pipe was erected thereon. No further anchoring of the pipe was found necessary, its great weight keeping it in place.

A single line of this pipe is sufficient for present purposes. It connects up with one of the above-mentioned 48-in. outlet pipes directly in front of the dam, and joins on to the steel pressure pipes at a distance of 7150 ft. from the dam. Its alignment closely follows the contour of the hillside, with all its curvatures, the minimum radius used being 150 ft. The pipe, as mentioned above, descends on a uniform grade, with a total fall of 42 ft. It is connected up to the two pressure pipes, leading down the spur to the turbines of the power station, by means of an intermediate drum-shaped steel header (Plate VIII.), which serves as a sort of buffer, or safety valve, between wood pipe and pressure pipes.

This wooden pipe is much cheaper than a metal pipe would have been, and, as far as life is concerned, the indications are that it will last just as long. It was certainly much easier and cheaper to instal at such an out-of-the-way place. As against an open channel, or flume, the loss of water will be very much less. There are no leaks whatever. An important point is that only such water is drawn from the source, through the medium of the pipe, as is actually required by the turbines, and none is wasted. Another point is that the wood pipe is always clear inside, and unobstucted, whereas open channels are subject to falls-in and disturbances from *débris*. The pipe is not subject to repairs like a wooden race or flume. Finally, the pipe enables every foot of head between the lake mouth and the lower termini of the pressure pipes to be utilized.

This wooden conduit line is of the continuous-stave construction. It is formed of 29 staves of different lengths, around the circumference (Plate VIII.), and has an internal diameter of 4 ft. The staves consist of well-seasoned Douglas fir, imported from America, free from blemishes, $1\frac{3}{4}$ in. thick by 6 in. wide, machined all over, with a V-grooved side joint, and a butt joint at the ends, where a thin strip of metal is driven in, half of it into each stave. The staves are held in place by mild-steel bands of various diameters, encircling the pipe—viz., $\frac{5}{8}$-in., $\frac{3}{4}$-in., and $\frac{7}{8}$-in.—so spaced as to give an even factor of safety of 5 throughout the

pipe, and capable of being tightened up, or loosened, by means of screw ends. These bands are placed only a few inches apart, never more than 9 in. The structure is thus practically a tube made of separate steel bands, or circular bolts, with a continuous wooden wall-filling. The staves are purposely kept quite thin, as the safety of the pipe depends not upon the thickness and strength of the wood, but upon the number and size of the steel bands, and the thinness of the staves contributes to the life or preservation of the wood. It is expected that the wood will permit of the transpiration of a small amount of water, which keeps the fibre in a healthy state. Heavy, stout thicknesses of staves would lead to decay. The staves were all faced by a special machine driven by the temporary hydro-electric plant at the power-house site, and the staves and bands delivered into position off the high-level tram. The pipe was thus put together in place, stave by stave, and working from both ends. It was erected by the Australian Wood Pipe Co., of Sydney, under contract. Though a novel undertaking in these dimensions in this part of the world, it appears to be an unqualified success, and was quickly erected, as much as 200 ft. being laid in 8 hours.

Steel Header.—The steel header (Plate VIII.) forming the junction between the lower end of the wood pipe-line and the upper end of the steel pressure lines, is 6½ ft. internal diameter and 15½ ft. long. It has 6 openings for the following purposes, viz. :—One 4-ft. opening at the end, which is the inlet for the wood stave pipe ; one 29¾-in. opening at the opposite end, for the scour pipe, which is ordinarily closed by a valve ; three 29¾-in. branches, on the lower side, being outlets for the steel pipes leading the water to the power station, of which outlets, however, only two are used at present ; one 4-ft. branch, on the upper side, for the vent pipe, for safety purposes. This vent pipe runs 150 ft. further up the hill, and terminates at an altitude a few feet higher than lake level. Its function is to act as a vent, or overflow device, in case the water in the pipe system is suddenly cut off at the power station. In a closed circuit a very severe shock, possibly leading to breakage, would be communicated to the wood and steel pipes.

This safety pipe is also a wooden one, of the same construction exactly as the main conduit, except that, for purposes of test, local King William pine has been used.

The concrete anchorages securing the header are bedded in solid rock, and are of a most substantial character. The sand and stone for making the concrete, obtained alongside of the steel pipe-line, were surface detrital matter, mostly conglomerate rock. It was thoroughly washed, and made an excellent concrete.

Steel Pipe-Lines and Distributing Pipes.—The two steel pipe-lines, or pressure columns (Plate VIII.), are laid abreast at the same level, down the steep incline of the mountain spur already mentioned, and connect the header with the power-station. The total length, measured on the slope, is 2993 ft., and the total static head available, together with that in the wood-stave pipe and the lake storage, is 1100 ft. Each steel pressure pipe has a 29¾-in. sluice valve near its junction with the steel header, and this can either be operated direct by hand or from the switchboard at the power station, in case of emergency. For mounting in place the sections of the pipe were simply rolled off the haulage-line trucks into position. Numerous heavy anchorages of cement concrete were built into saw-toothed excavations in the rock-surface at every change of grade or direction of the pipes, and completely surround the latter at such places, and, in addition, there is a masonry support near each joint, taking both pipes. The tendency to creep is resisted by the anchorages, the pipes being given ample bearing upon the concrete by cast-iron thrust rings embedded in the same. The total number of cubic yards of concrete in the anchorages amounted to 695 yds. Of this, 58 % was made up with concrete, and the balance (42 %) of displacers of conglomerate rock. Cement used was 0.88 of a cask per cubic yard of concrete.

The double steel-pipe column is in three sections, with diminishing internal diameters, running from 29¾ in. at the top, through 26 in. at the centre, to 22 in. at the bottom, and with an increasing wall-thickness of 7 mm. at the top to 14 mm. at the bottom. The metal was specified to be Siemens–Martin firebox

steel plate, having a tensile stress of not less than 25 tons and not more than 29 tons per square inch, with a minimum elongation of 20 % in a length of 8 in. and a limit of elasticity of not more than 14 tons per square inch, the welded joint not to have a less strength than 90 % full plate. Hot and cold bending tests were also required.

After considerable inquiry into the various forms of joint in use, a decision was made in favour of a welded pipe with patent muff joint, made by the Ferrum Co. The joint is very simple, and has been used most extensively in many high-pressure plants. The pipes were about 19 ft. 8½ in. long, and each is made from one plate with one longitudinal weld. In the case of bends there is also a circumferential weld.

Before leaving the manufacturer the pipes were coated inside and out with a preservative compound, and after erection, jointing, and testing received another coat of the same compound.

Each pipe was tested at the works to 1½ times its maximum working pressure. At anchorages and other special points a flange joint was employed, the ends of the pipe being flanged up inside two heavy flange rings. With these exceptions, the whole line is jointed with muff joints.

The muff joint (Plate VIII.) is a form of spigot and faucet joint, with joint rings for securing the packing. The packing is made of strands of hemp rope treated with a special preparation to give it hardness. It is caulked in with ordinary caulking tools, strand by strand, and great care has to be exercised that the material is put in solidly. The packing ring is then put up against the packing and the bolts tightened.

Only one joint out of the whole length of pipes had to be re-made; the few other leaks were taken up by tightening the bolts on the packing rings.

In order to provide means for inspection of pipe, a sleeve pipe (Plate VIII.) has been inserted near the anchorages. This pipe can be lowered downhill sufficiently to admit a man through the opening.

The erection of the pipes was commenced at the power-house bottom anchorage and worked from there up the hill. They were laid hard up against one another, spigot to socket, up to the next anchorage, and the closing pipe cut to the correct length, the cutting being done in the company's workshop before being sent out, so as to avoid cutting them by hand. In order to expedite the work temporary supports were placed under the pipes during erection, and, on reaching an anchorage, the concrete forming the anchorage was placed around the pipe before proceeding further. No expansion joints were required for the pipe-line, as each muff joint serves this purpose.

At the lower terminals at the power-house the two steel pipes change their direction and become solid, welded, flanged distributing pipes, 22 in. diam. (Plate IX.) They continue parallel to the long wall of the building, one pipe extending to the most distant turbine, with a thrust block at its end, the other stopping about half way, and being joined to the other pipe by means of a connecting casting, similarly provided with a thrust block. This casting, in conjunction with certain valves (Plate V.), allows of either pipe being used to supply any one of the four turbines with pressure water.

In the line of distributing pipes are four cast-steel tee pieces, one leading into each turbine. Between this piece and the turbine is an adjustable joint, which was helpful in erecting the distributing pipe, and will also be of service should it be required to remove the turbine inlet stop valve at any time. As the pipes, valves, etc., at this point are subjected to the greatest pressure, all the bends, branches, etc., are very massively anchored to give the greatest security should shocks occur. All this part of the work is built on solid rock.

The main control valves are 22 in. in diameter and three in number, the third one being a safeguard, in this sense, that it would be possible to operate at least two large turbines in case of an accident to the distributing pipes. They are operated by an hydraulic cylinder, working from the pressure column, which, for its part, is controlled by a pilot valve, worked either by hand

direct, or by means of a small electric motor put into motion from the power-station switchboard, in a similar manner to the other valves referred to above as controllable in various positions distant from the switchboard. These valves are fitted with ample by-pass valves, which are intended for hand control only. It may be stated that it is advisable to only use the electric control in case of emergency, as the electrical operation under full pressure throws a heavy strain on the valves. The valves are arranged to close very slowly—in about 10 seconds—to avoid dangerous rises of pressure in the pipes.

Other accessories on the distributing pipes are 10-in. valves and branches leading to the exciters, the same being in duplicate from each distributing pipe, and a scour valve for emptying the pipe-line, leading into the river alongside.

In order to keep a continuous record of the amount of water used, a Venturi meter is fixed to each of the distributing pipes, and, in addition, to the consumption per minute, this shows the total gallons per day, and also gives a diagrammatic representation of the momentary fluctuations in the supply. The makers of the Venturi meters guarantee their accuracy within 1 % of actual flow, and the friction loss 2′ 0″ at full load.

POWER-HOUSE.

The power-house consists of a spacious and lofty building, 107 ft. 8 in. in length by 40 ft. 6 in. in width, and 29 ft. in height to the eaves. It has, on the river side, an annex for housing the valves and distributing pipes, 15 ft. 6 in. wide by 16 ft. 11 in. long, and another annex on the opposite side, 13 ft. 4 in. wide by 65 ft. long, containing the switch rooms, offices, stores, &c. The roof is of corrugated galvanized iron, lined with fibro-cement sheets, and is carried by steel trusses. The foundations for the building and turbines are on solid rock, the main tail-race running longitudinally with the building. The spaces to accommodate the turbine foundations were blasted out as small as possible, to economize on concrete, after which forms

for the whole of the plant foundations were dropped into place and accurately set up, and the concrete then filled round them. A good deal of time was saved by this method over the ordinary way of fixing up one foundation after the other, using the same form successively. The tail race is lined throughout with concrete, and has a fall of 1 in 70 to the river. The yardage of concrete used for foundations was 1230, and the casks of cement used 993, or about 0.8 cask of cement to the cubic yard. The walls of the building are made of concrete, having large, cast-iron frame windows, giving ample light. The total amount of concrete in the walls, floors, etc., was 770 cub. yds., and the quantity of cement used 1205 casks. A special material for hardening the surface of the cement floor was used, to avoid abrasion of same. Concrete, reinforced with expanded metal, was employed to form the floors of the switch and lightning arrester rooms.

For the purpose of handling the heavy lifts, a 10-ton hand-operated crane is installed, running along the entire length of the building.

TURBINES, GENERATORS, ETC.

Four subsidiary pipe branches are laid at right angles off the main distributing pipes from the aforementioned tee pieces, one to each turbine, through a hydraulically-controlled stop valve with a by-pass, similar to the 22-in. valves, operated by hand and from the switchboard. Each branch terminates in a nozzle, $4\frac{5}{8}$ in. diam., extending into the turbine casing itself. There are four turbines, all of similar construction. They are of the Pelton wheel impulse turbine type (Plates X., XI., and XII.), and consist, in the main, of a large, solid, steel disc, machined all over, each mounted on a heavy steel shaft, and surrounded by a shell-like cast-iron casing. On to the circumference of each wheel are bolted heavy double spoon-shaped buckets, into which the jet of water is accurately directed from the nozzle. These spoon-shaped buckets are made of the finest cast steel obtainable, and are machined on the inner bowl-shaped faces as smooth as a mirror. Where the two bowls touch each other is a cutting edge,

ground as fine as a knife blade, and it is on this edge that the jet of water strikes. The jet of water is thus divided into two parts, striking into and filling each bowl, and pushing the wheel forward in the direction of the motion of the water. The buckets are arranged at such intervals on the circumference of the wheel that, by the time the impact of the water has exhausted itself in one bucket, the next following is taking the stream. At the same time the construction is such that no shock is communicated to the circumference of the wheel by the jet. The force of the jet, notwithstanding the great pressure behind it, is completely lost out of the water by the time the latter leaves the bowls of the bucket. The shaft of the turbine wheel itself runs in two massive bearings, lined with high-grade white metal and bolted to a bed-plate. The whole wheel, complete with shaft, is carefully balanced. Water traps of a simple and efficient design are provided where the shaft goes through the casing. The spent water drops into the underground tail-race running longitudinally underneath the row of turbines, and the water falls direct into the stream, thus obviating wear of the floor. A steel plate lining for turbine pit is bolted to the under side of the bed-plate, and forms a water-tight protection for foundations.

The controlling of the water is by Boving's patent governor, which is a combination of deflector and needle regulation, both being worked automatically by the same servo-motor and pendulum (which is the centrifugal element), and both comprising what may be termed the governor. The deflector is placed above the jet, and is designed to deflect the whole jet away from the wheel when the deflector edge reaches to the centre line of the jet. This deflector is positively connected to the servo-motor shaft, and to each position of servo-motor and deflector corresponds a certain output—that is, a certain jet diameter. The lever connections are so arranged that in normal operation at all loads the deflector stands just clear of the jet, ready to come into operation, without any dead motion, which is of the greatest importance for close regulation. This deflector diverts the full jet in from $1\frac{1}{2}$ to 1 second. The connection between the needle

and the servo-motor shaft is made through an oil dash-pot with a long slotted link (Plate XIII.)

When load is thrown off suddenly the governor instantaneously brings the deflector into the jet, thus diverting the water and reducing the output without affecting at all the flow of water in the pipe-line. The needle, which has always the tendency to close when not checked by the governor, by reason of a compression spring mounted on the stem, follows quite slowly the movement of the governor, reducing the flow of water to adapt it to the new conditions after the change. The closing time of the needle can be adjusted between 15 and 30 seconds by means of a regulating valve in the by-pass through which the oil escapes from the dash-pot cylinder.

When load is thrown on the governor lifts the deflector out of the way, and, at the same time, the needle opens quickly, the dash-pot offering no resistance.

The pendulum, or the centrifugal part of the governor, is driven from the turbine shaft by a double belt. The rotating balls of the pendulum are connected by the pendulum sleeve, and a system of levers, with a small pilot valve, which controls the high-pressure oil to the piston of the servo-motor differential cylinder. In order to keep a steady speed at all loads, or any speed drop between no load and full load up to 6 % or 7 %, a return gear, connecting the servo-motor shaft to the pendulum sleeve and a compensating device, is provided. The adjustment of this gear can be altered during operation.

The speed can be adjusted during operation either by means of a small hand-wheel on the return gear spindle or from the switchboard by means of a small direct-current motor.

The hand-regulating gear on the governor can be readily connected, or disconnected, by pulling a lever, the automatic regulation being at the same time cut off or thrown on.

A rotary-type oil pump provides high-pressure oil at about 220 lb. pressure for the governor, and is connected on the suction side with a low-pressure oil-chamber in the governor casing. The bottom compartment of the governor forms a pressure accumu-

lator with air cushion, and is provided with an overflow valve through which the oil flows back to the low-pressure chamber after passing through cooling pipes immersed in the tail-race.

Means for access are provided for inspecting the nozzle, deflector, etc. The parts under the floor level are covered in with cast-iron chequered plates, giving the whole a very neat appearance.

The turbines themselves are designed for 500 r.p.m., with a normal rating of 1750 b.h.p. each, and have a test efficiency of 84.6 %.

Each turbine is direct-coupled, by means of a flexible coupling, to an electric generator having a rated capacity of 1200 kw. at .8 power factor, and this generates three-phase current at a voltage of 6600, with a periodicity of 50 cycles per second. The test efficiency of the generators is 95 %, and their construction is heavy and liberal in all dimensions, the same as the turbines.

For field current of main generators three small turbine-driven exciter sets of 50 kw. each are used, which are of nearly similar design to the main turbo-generators, except that the deflector and needle dash-pot are dispensed with, the governor operating the needle directly. These sets run at a speed of 1000 r.p.m.

Three of the main generating sets, fully loaded, are required to supply the present needs of the mines and works for power. The fourth machine is a stand-by for emergencies.

The leads throughout the installation are run through conduits embedded in the concrete floors, which makes a very good job electrically, and is also sightly in appearance.

The various electrical units are controlled from a large marble switchboard, mounted on an elevated gallery overlooking the main hall of the power-house. The switchboard has the following control panels: — Three exciter machine control panels, one Tirrill regulator panel, four main generator control panels, two outgoing transmission line panels, one local power and lighting panel, two valve control panels.

The switches are mechanically operated by hand from the switchboard, but are placed some distance behind it, so that no

dangerous high-tension current is brought to the operating side of the switchboard, thus ensuring the maximum of safety to the attendant.

The whole of the switch gear, etc., is contained in a three-story annex of the main building. The lower story houses the current and potential transformers for the switchboard instruments, the local power transformer, and the light transformer. The middle story accommodates the main oil switches, each set in a separate cubicle, or cell, for motives of safety. The top floor provides room for the lightning arrester equipment. All interconnecting cables are carried through conduits buried beneath the floor.

The pressure pipes, valves, and turbines were supplied by Messrs. Boving and Co., of London, and the entire electrical equipment by the General Electric Co. of the U.S.A. through their agents, the Australian General Electric Co., of Melbourne.

TRANSMISSION LINE.

The transmission line forms the link conveying the power from the above generating station to the works and mines. It is a little less than 5 miles long, and runs in a practically straight line across country to the reduction works. It is a double line, for two systems of three conductors each are laid parallel to each other 50 ft. apart, the duplication being for the sake of security of operations and convenience in making repairs. The poles are round timber, of celery pine, locally obtained, and are spaced an average of 130 ft. apart. The ground traversed has been cleared of timber for protection against fire. Each pole is fitted with two cross-arms, with stays, and carries the usual crown-glazed high-tension porcelain insulators.

The six conductors themselves consist of 19/11 standard copper cables, and each side of the line is provided with the electrolytic type of lightning arresters, in duplicate, at each end—*i.e.*, at generating and substation.

SUBSTATION.

The substation itself is built of concrete, with steel roof, in the same manner as the power-house, and comprises a main hall

and an annex for the auxiliary gear. The main substation switchboard is situated on the ground floor of the hall, and controls the incoming 6000-volt transmission line to and from the transformers, and distributes the 3000-volt current to the Mount Lyell and North Mount Lyell mines, also to the blowing plant at the reduction works, besides supplying current to other transformers for the 550-volt low-tension distribution board. The switch gear is of similar design to that in the power-house, and the lightning arrester equipment is located in the gallery above the high-tension switchboard.

The low-tension switchboard is one which had been used for several years at the steam-generating station at the reduction works. It was simply remodelled so as to do service as a distributing switchboard controlling a number of minor circuits throughout the various works. Another switchboard, formerly used in the company's first electrical installation, controls the battery and all direct-current circuits. The storage battery supplies temporary lighting service should the other plant be shut down for any reason, and it is also used for actuating the automatic trips of the switches.

Two 75 kw. motor generators are located in the main hall for general lighting, and a small Pelton wheel impulse turbine is being installed as an alternative drive for these generators. It will be worked off the smelter water-supply main, which runs close to the substation. This is for the purpose of a separate supply of electric light, in case of a complete shut-down of the main power scheme. The current for the electric crane in the new converter plant is supplied by two motor generators delivering direct current at 500 volts.

The substation annex contains the main transformers. These are of the single-phase, oil-insulated, water-cooled type, nine in number, and of 700 kw. each, arranged in banks of three. Two banks are in constant use, and the third is kept in reserve. All the high tension equipment was also supplied by the General Electric Co. of the U.S.A.

POWER APPLICATIONS.

In order to obtain the maximum advantage derivable from the change-over from steam-driven electric power to hydro-electric power, it was necessary to instal electrically-driven blowing engines for the smelter furnaces and converters. This machinery is housed in a substantial structure of concrete and steel, situated on the slag-dump level of the reduction works, in close proximity to the smelting plant. It is connected with the sub-station by two 3000-volt circuits. These circuits drive six motor-driven centrifugal blowing engines through suitable control-panels on the switchboards in the engine room. The design of the blowers themselves is an adaptation of the multi-stage centrifugal pump design used for water, but here applied to the compression of air. It represents the latest advance in air-compression. The furnace blowers each have a capacity of 25,000 cub. ft. free air per minute, at pressures varying from 4 lb. to 6 lb. per sq. in., and their driving motors are 920 b.h.p each. They are ventilated by separate motor-fan sets, one on each blower. The blowers themselves are of the double-ended balanced type, having three impellers on each end, and take the air in at both ends, delivering it into the middle wheel chamber. Each end of the blower deals with half the quantity of air to be compressed, and by this means no balance device is required. These blowers on tests showed an adiabatic efficiency of 75 %, which is very satisfactory.

The converter blowers are of similar construction, and have 10 impeller wheels in series. They have a capacity of 3500 cub. ft. of free air per minute, at 12 lb. pressure, and are driven by motors of 250 b.h.p. In these blowers the air inlet is at one end and the discharge is at the other end, and an effective balancing device is provided. The adiabatic efficiency is 73 %.

All of the various blowing engines run at a speed of 3000 r.p.m., and are fitted with tachometers and pressure gauges. The bearings are oil-ring lubricated and water-cooled. The blowers are mounted on high and massive concrete pedestals, giving access to the various piping for the inlet and the discharge of air, and

the electric connections, etc. The air is drawn from the atmosphere at an elevation above the engine-room level, through three outside towers, the intake openings being guarded with screens.

The whole of the blowers and switch gear complete were manufactured by Messrs. Brown, Boveri and Co., of Baden, Switzerland.

Since the completion of all the above works, it may fairly be said that, from an operating point of view, the plant has given every satisfaction, and the scheme as a whole may be called an unqualified success. It has most efficiently and economically replaced the steam-driven electric plant formerly in use.

The diminution in expenditure for power-generation under the new scheme is most marked, and the saving runs into a large sum per annum.

The outlay for maintenance promises to be very slight, and, while there is some danger of interruption, experience has shown, up to date, that this is not likely to be calamitous at any time, the duration of interruptions due to causes inherent in the electrical portion of the system being limited to short periods, such as a few minutes to an hour, or a couple of hours at the most. There has, therefore, been no necessity for retaining any portion of the old steam plant as a stand-by for the sake of emergencies, even in the beginning. The change-over from steam to water-power was made, without trouble or cessation of operations, during the last days of November, 1914.

Briefly summarizing the various purposes for which the Lake Margaret hydro-electric scheme is supplying energy for power and light for the company's requirements, the following principal points of application may be enumerated, together with the purposes to which the energy is applied :—

Mount Lyell mine, for hauling, winding, pumping, air-compression, lighting, machine shops, etc.

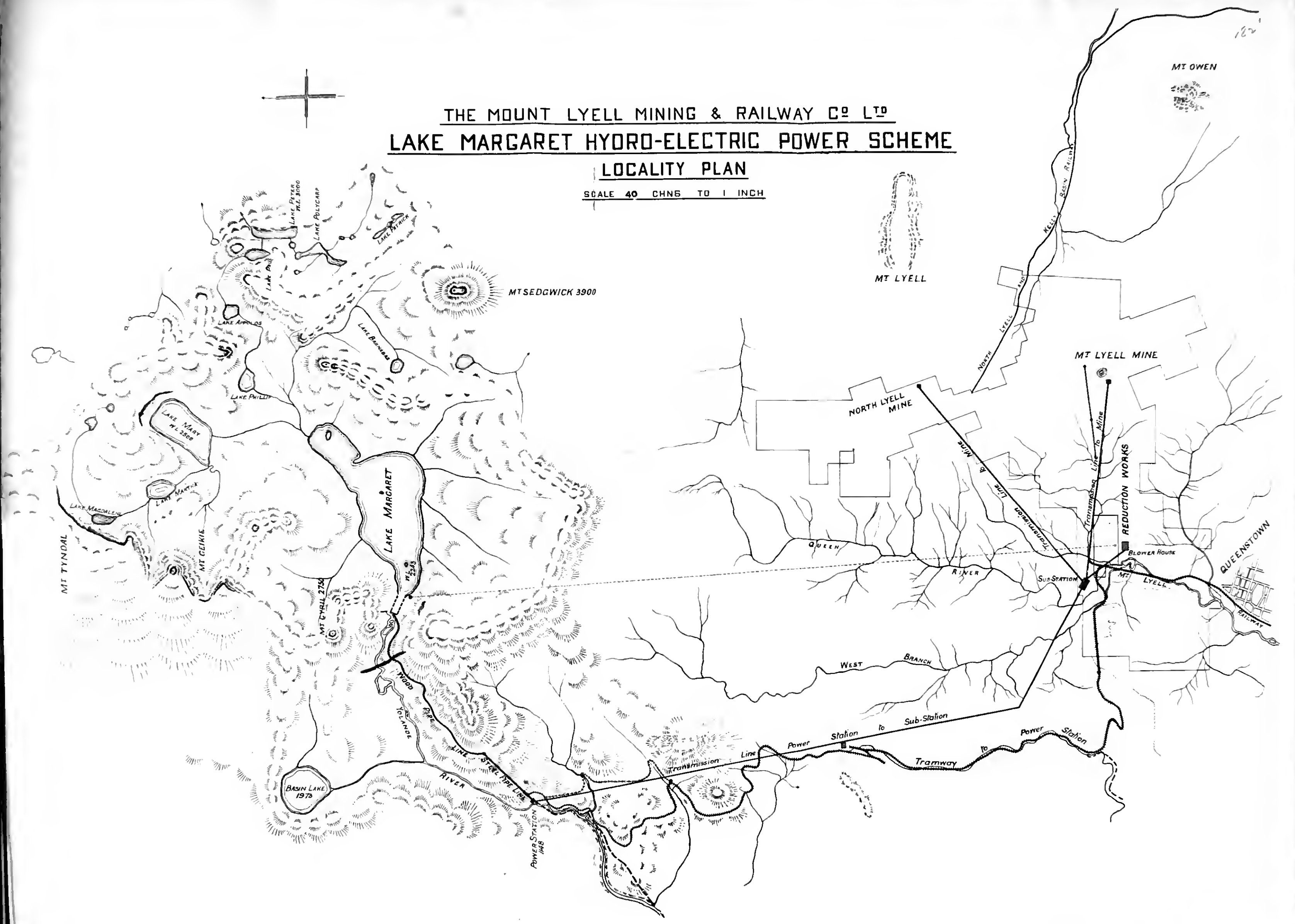
North Mount Lyell mine, for hauling, winding, pumping, air-compression, and lighting.

Reduction works department, for air-compression, hoisting, haulage, general power, elevating granulated slag, and lighting.

Converter department, for air-compression, general power, and lighting.
Reduction works valley pumping plant, for water supply, if needed.
Reduction works and railway department workshops, for general power and lighting.
Haulage lines, between reduction works and mines, for ore transmission over the intervening mountain saddle, also on Howard's Plains firewood haulage incline.
Township and works, for lighting generally.

The amount of power daily generated at the present time runs into 5000 h.p. The limits of the present scheme, however, are by no means reached by this figure, and, in addition, the water which is now utilized can again be used at a distance of about $1\frac{1}{2}$ miles further down the Yolande River, with an available fall of 400 ft., thus adding a further 40 % to the maximum power which can be derived from the present, or upper station.

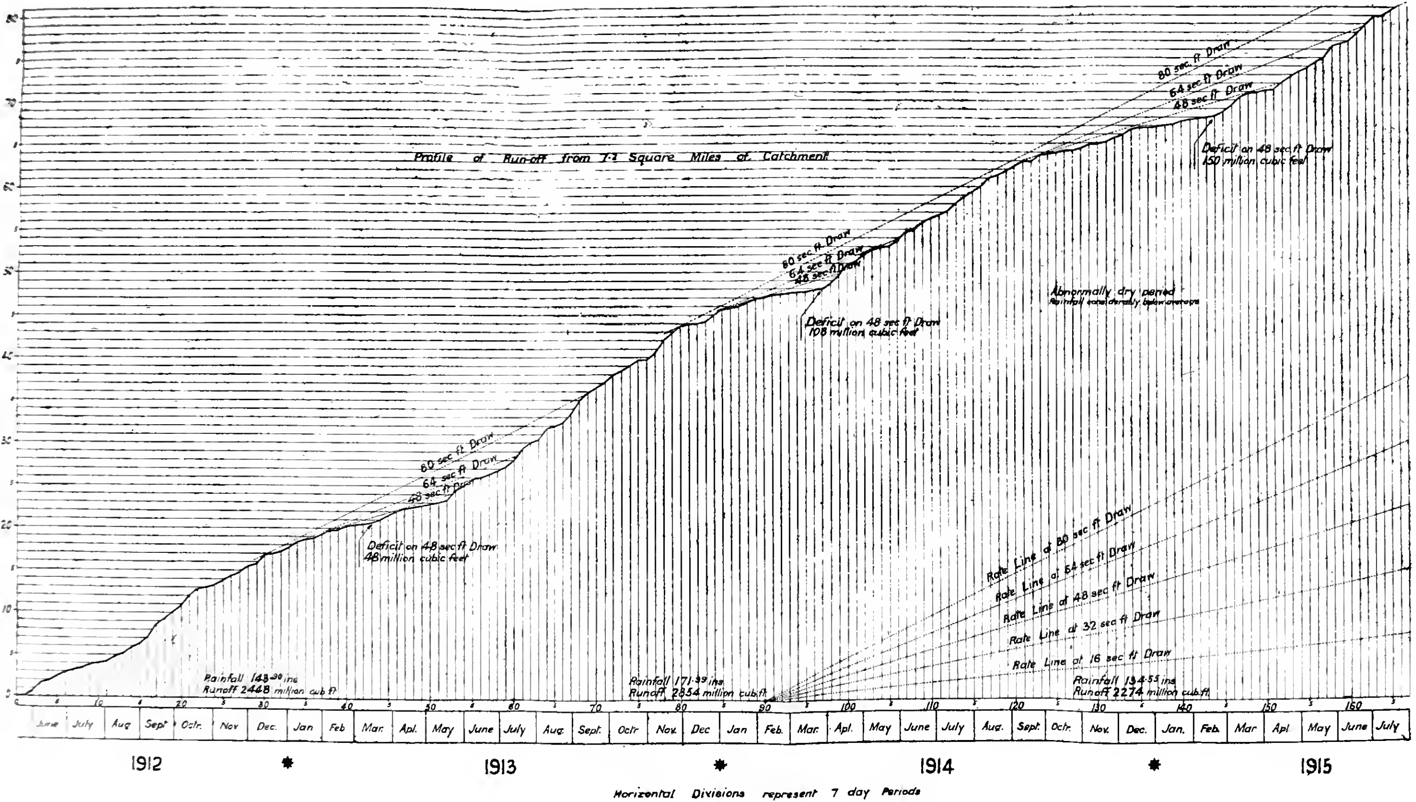

182'


THE MOUNT LYELL MINING & RAILWAY Co Ltd
LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME
LOCALITY PLAN
SCALE 40 CHNS TO 1 INCH
MT OWEN
MT LYELL
MT SEDGWICK 3900
MT LYELL MINE
NORTH LYELL MINE
LAKE MARY W.L. 2500
LAKE MARGARET
LAKE MAGDALENE
LAKE MARTHA
LAKE PHILLIP
LAKE PETER W.L. 3000
LAKE POLYCARP
LAKE PATRICK
MT TYNDAL
MT GEIKIE
MT CYRIL 2750
BASIN LAKE 1973
QUEEN RIVER
WEST BRANCH
REDUCTION WORKS
BLOWER HOUSE
SUB STATION
MT LYELL RAILWAY
QUEENSTOWN
NORTH LYELL
KELLY BASIN RAILWAY
Transmission Line to Mine
Transmission Line Power Station to Sub-Station
Tramway to Power Station
WOOD PIPE LINE
STEEL PIPE LINE
YOLANDE RIVER
POWER STATION 148

MOUNT LYELL MINING & RAILWAY Co Ltd

ARGARET HYDRO-ELECTRIC POWER SCHEME

RIPPL DIAGRAM.

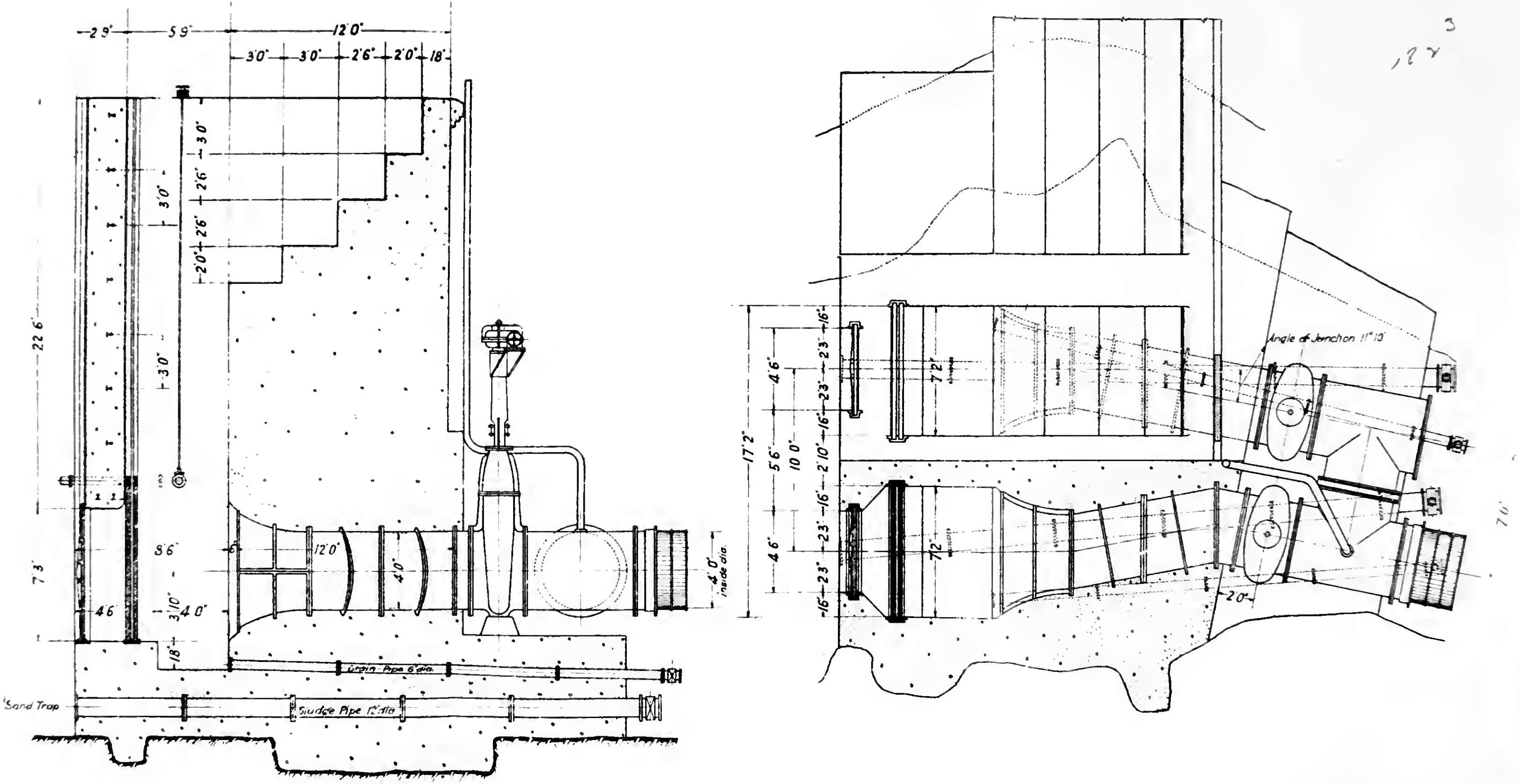


THE MOUNT LYELL MINING & RAILWAY C[o] L[td]

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME

RIPPL DIAGRAM.

182

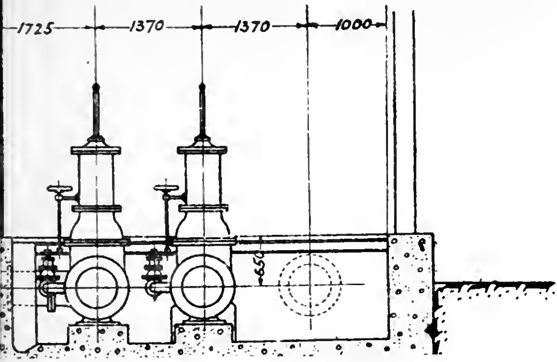


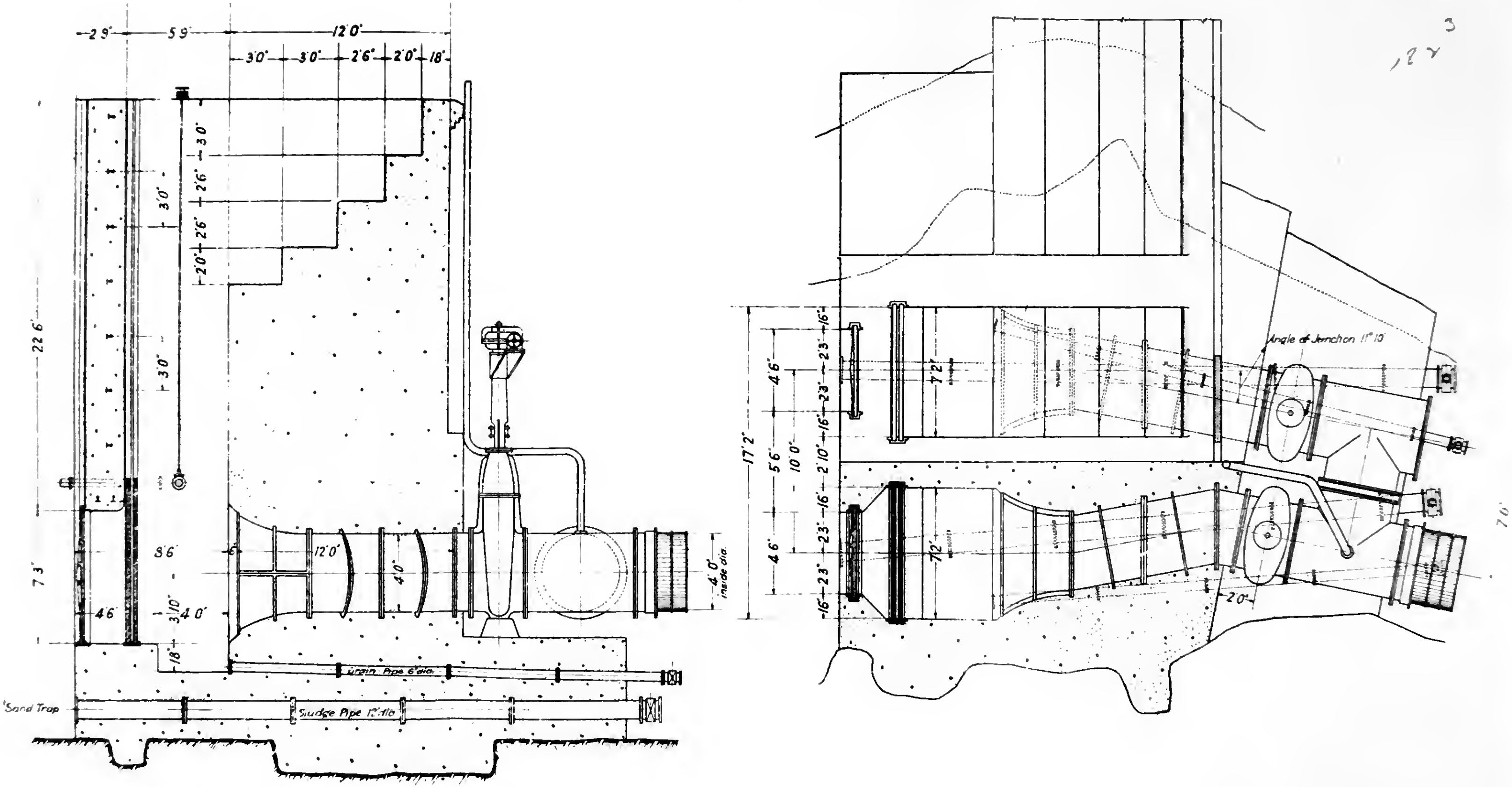
THE MOUNT LYELL MINING & RAILWAY C^O L^TD

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME—HEAD WORKS

SCALE—⅛ INCH - 1 FOOT.

182

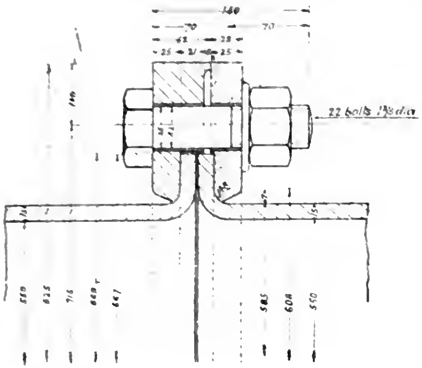




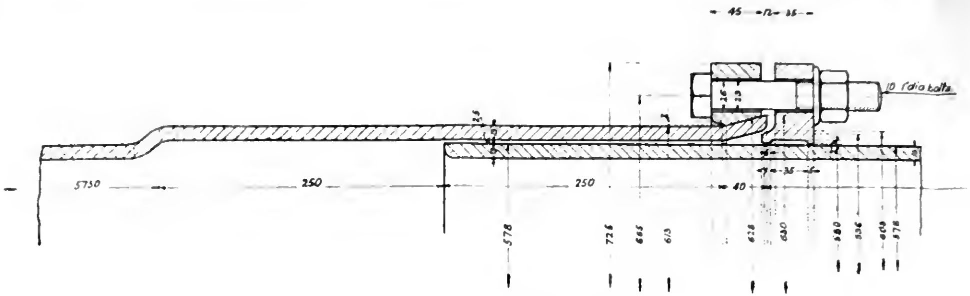
THE MOUNT LYELL MINING & RAILWAY Cº Lᵀᴰ

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME—HEAD WORKS

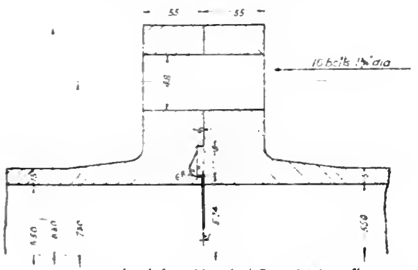
SCALE — ⅛ INCH - 1 FOOT.



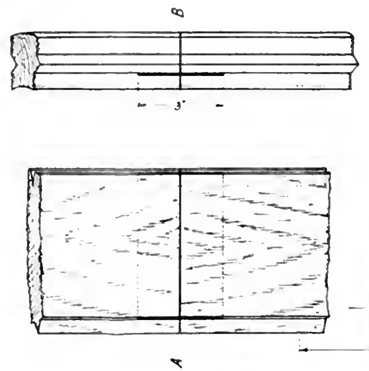
— Joint for Mild Steel Pipes (flanged) —



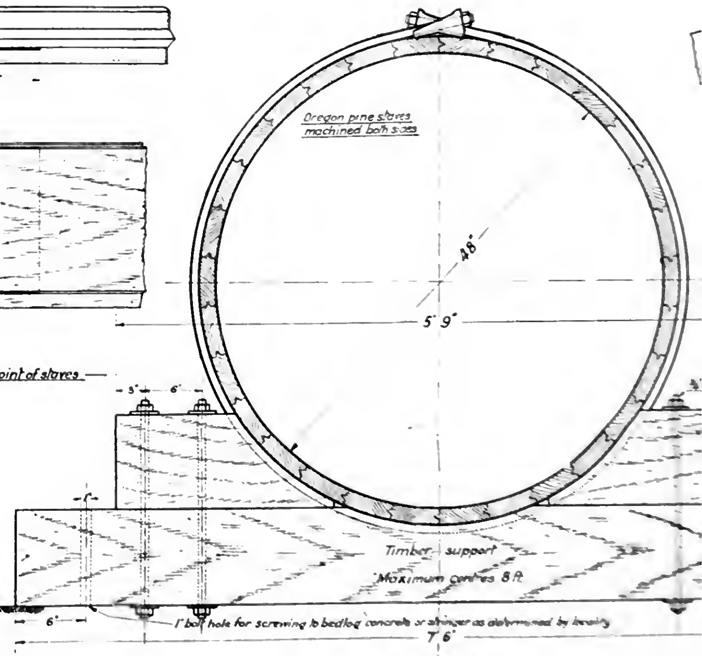
— Expansion Joint —



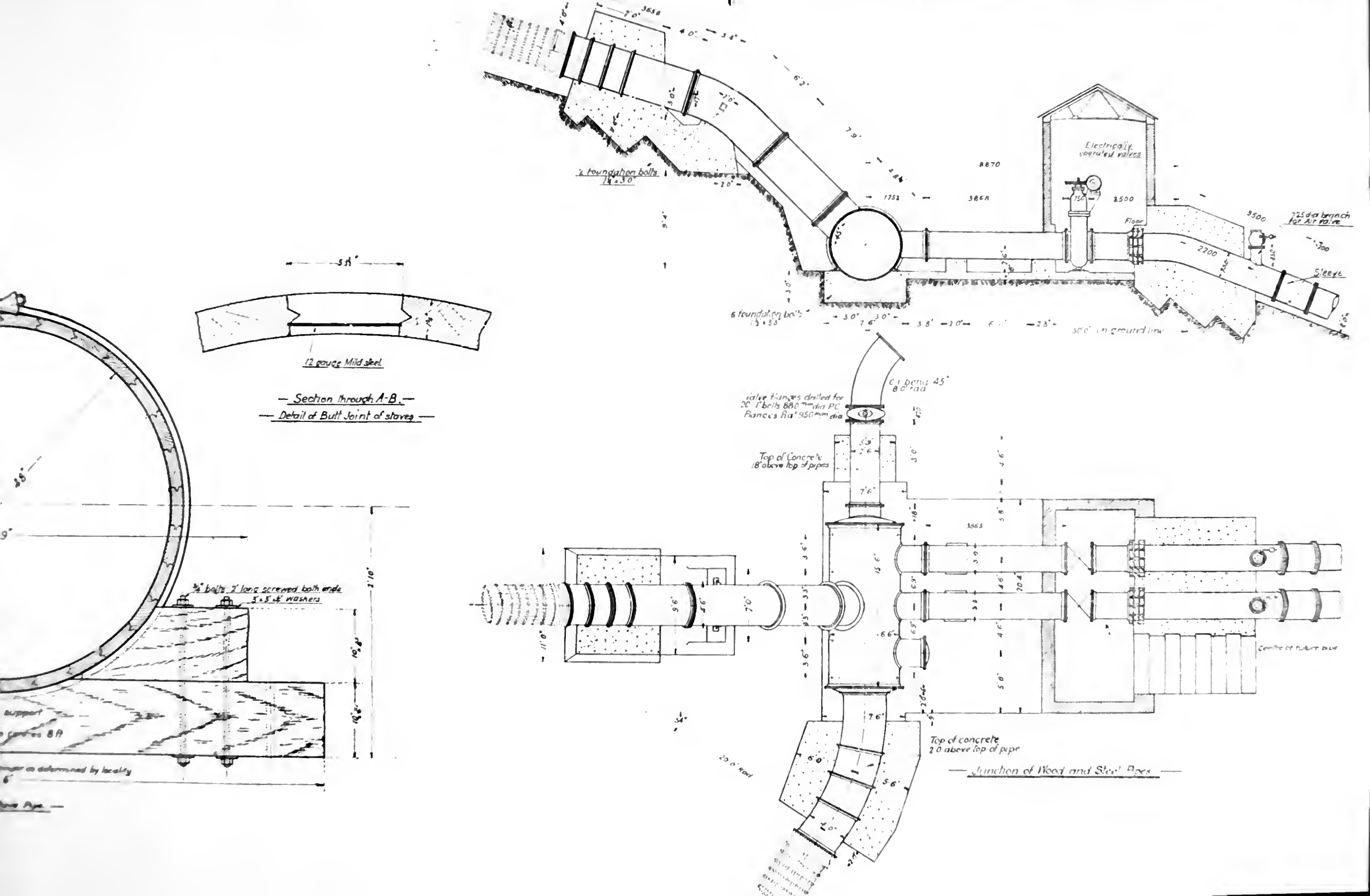
— Joint for Mild Steel Pipes (welded flange). —

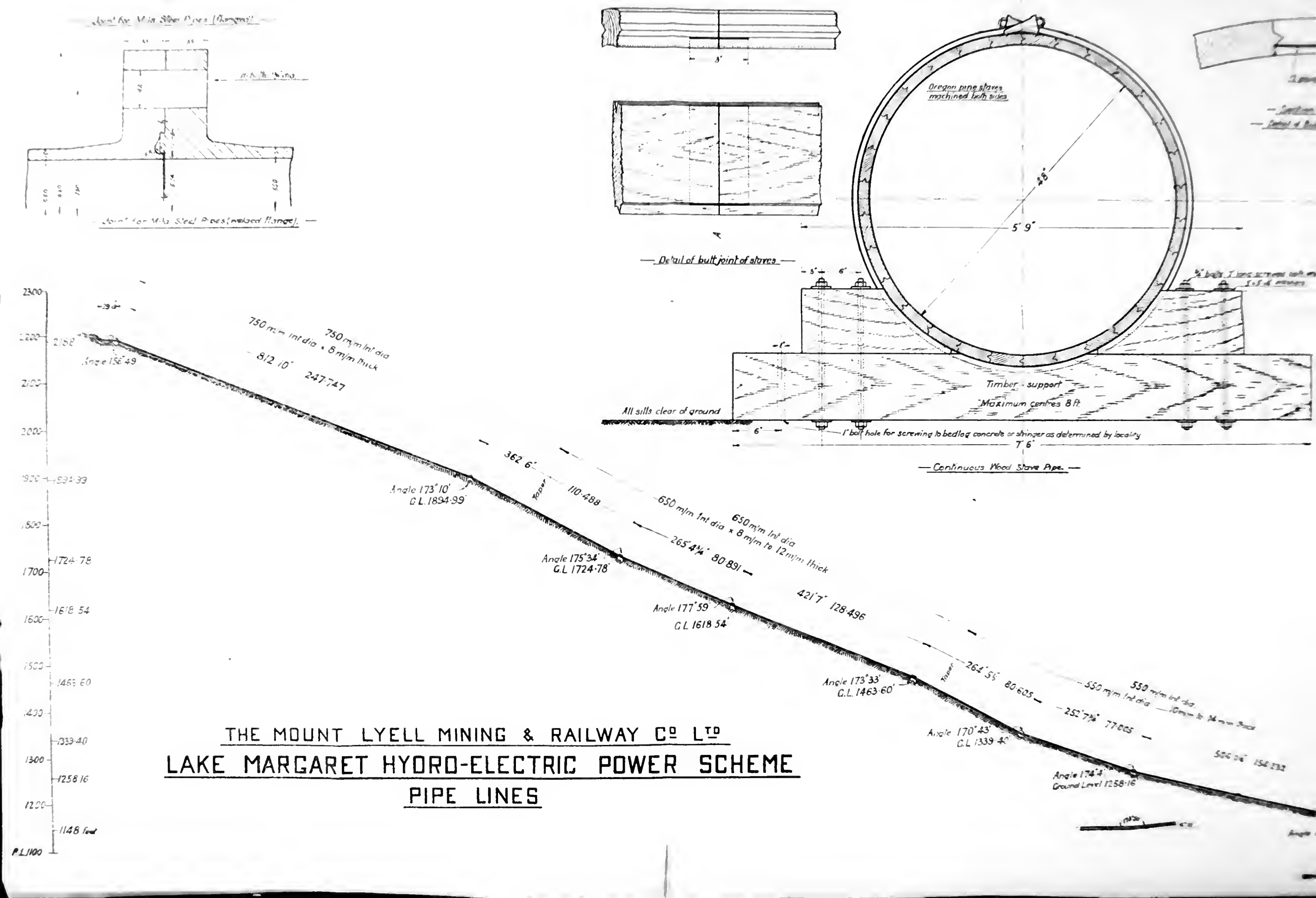


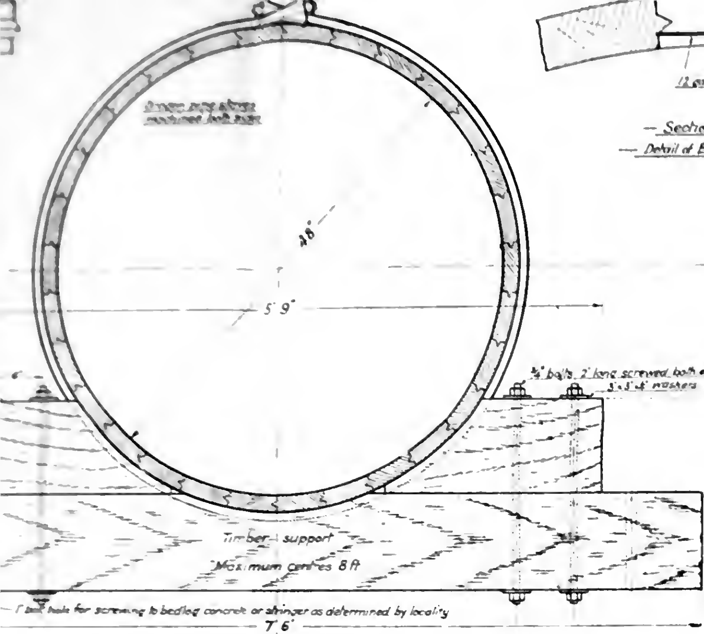
— Detail of butt joint of staves —



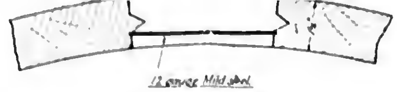
— Continuous Wood Stave Pipe —


Electrically operated valves
Floor
Sleeve
6 foundation bolts
12 gauge Mild steel
— Section through A-B. —
— Detail of Butt Joint of staves —
¾" bolts 2' long screwed both ends
Top of Concrete 18" above top of pipes
Top of concrete 2' 0" above top of pipe
— Junction of Wood and Steel Pipes —

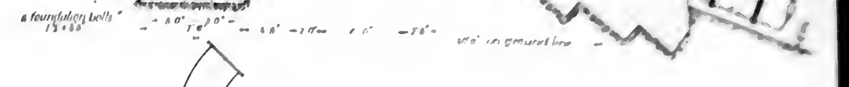

Joint for Mild Steel Pipes (welded flange).
Detail of butt joint of staves
Oregon pine staves machined both sides
4'8"
5' 9"
Timber - support
Maximum centres 8 ft
All sills clear of ground
1" bolt hole for screwing to bedlog concrete or stringer as determined by locality
7' 6"
Continuous Wood Stave Pipe.
750 m/m Int dia x 8 m/m thick
750 m/m Int dia
812' 10"
247.747
Angle 156° 49'
2168
Angle 173° 10'
G.L. 1894.99'
362' 6"
110.488
Taper
650 m/m Int dia x 8 m/m to 12 m/m thick
650 m/m Int dia
Angle 175° 34'
G.L 1724.78'
265' 4¼"
80.891
421' 7"
128.496
Angle 177° 59'
G.L 1618.54'
Angle 173° 33'
G.L. 1463.60'
264' 5½"
80.605
550 m/m Int dia
252' 7¾"
77.005
Angle 170° 43'
G.L. 1339.40'
Angle 174° 4'
Ground Level 1258.16'
2300
2200
2100
2000
1900
1894.99
1800
1724.78
1700
1618.54
1600
1500
1463.60
1400
1339.40
1300
1258.16
1200
1148 feet
P.L. 1100
THE MOUNT LYELL MINING & RAILWAY Co LTD
LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME
PIPE LINES

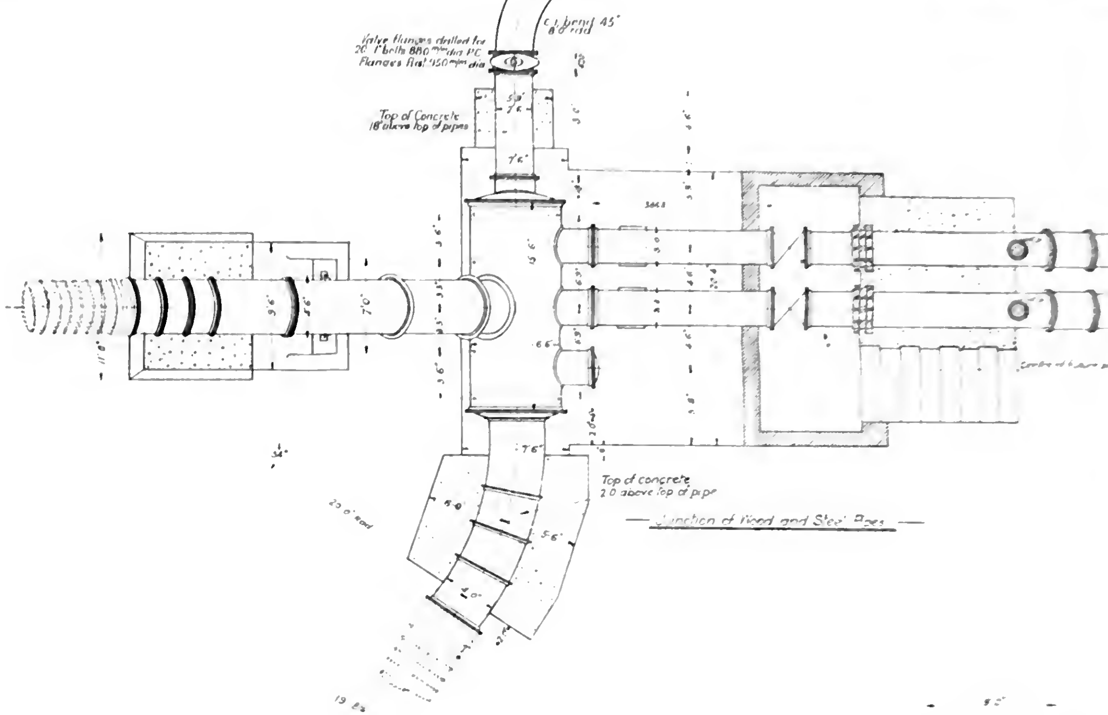


— Continuous Wood Stave Pipe —

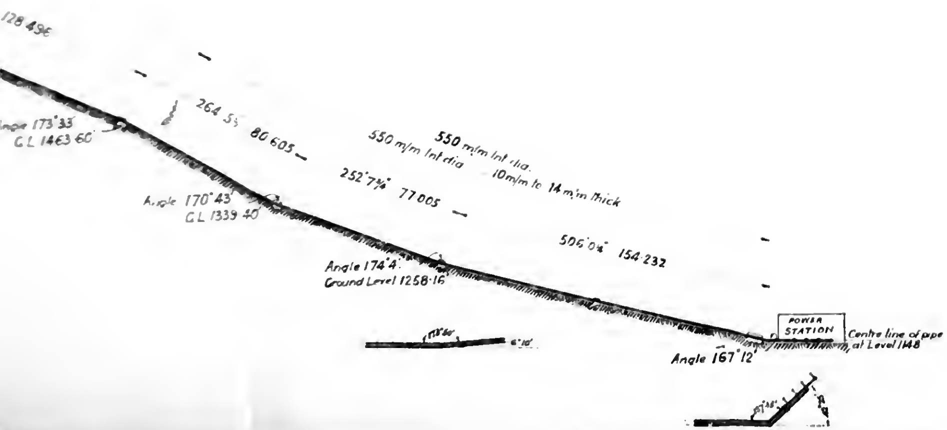


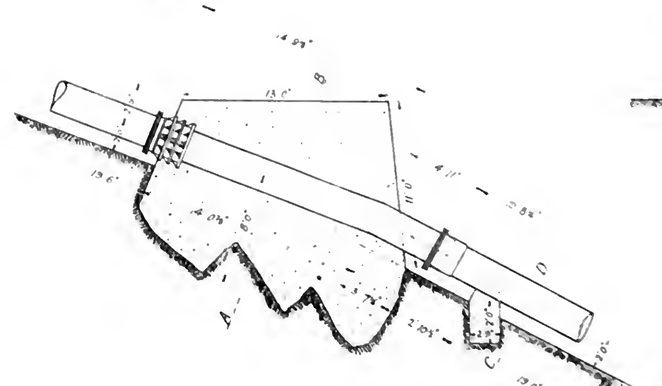
— Section through A-B. —
— Detail of Butt Joint of staves —



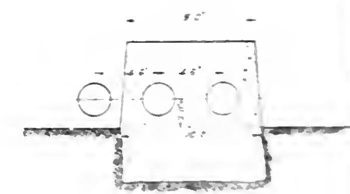


— Junction of Wood and Steel Pipes —



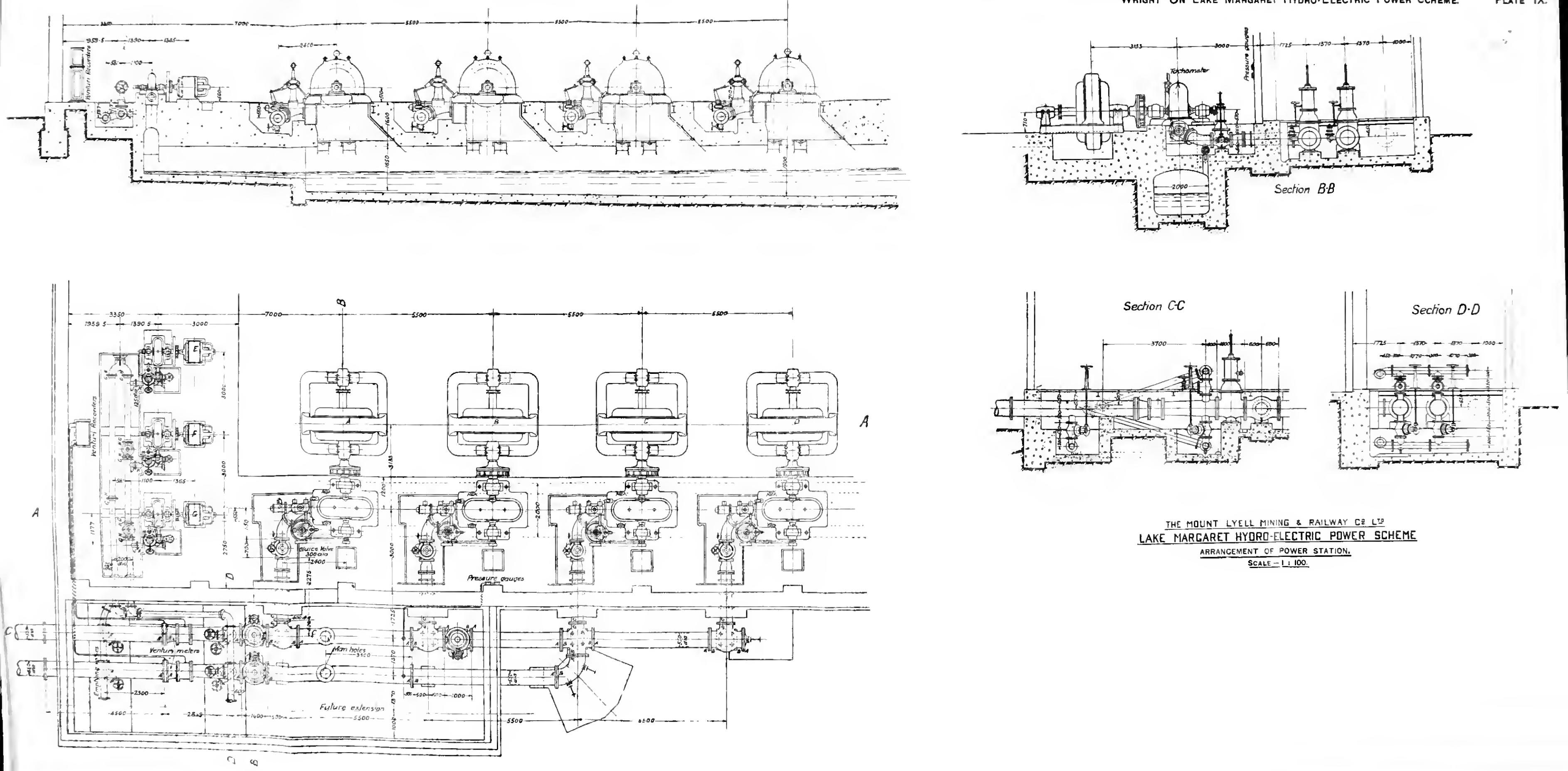


— Steel Pipe Line Anchorages —

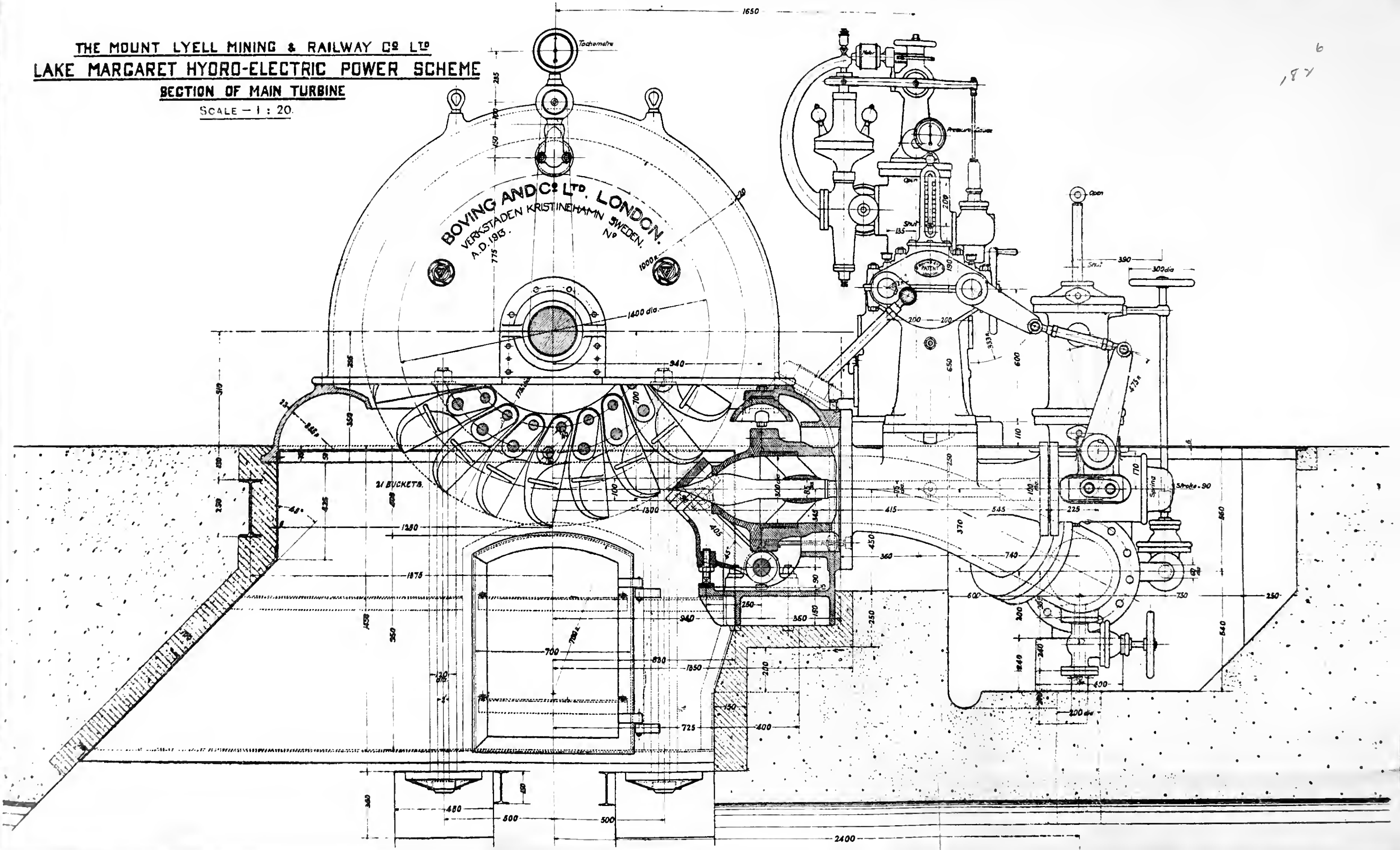


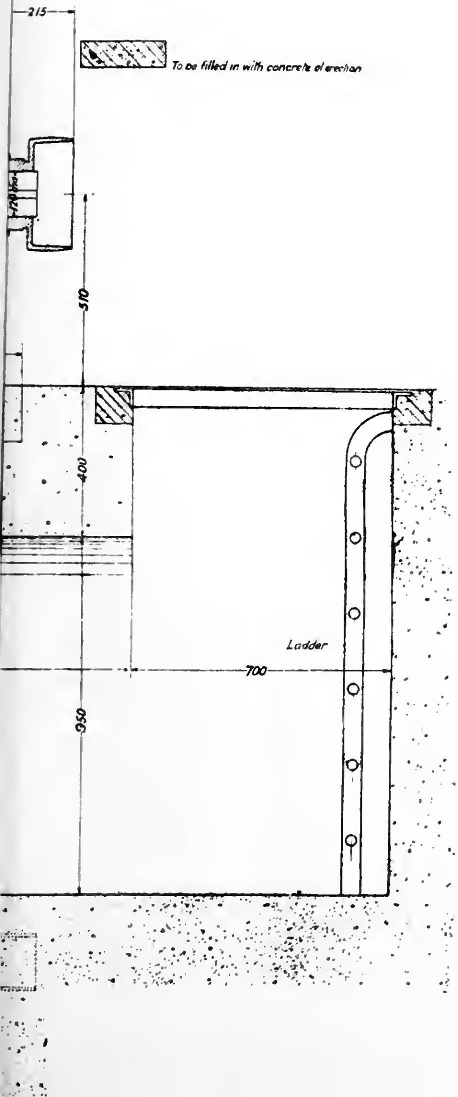
Section A-B

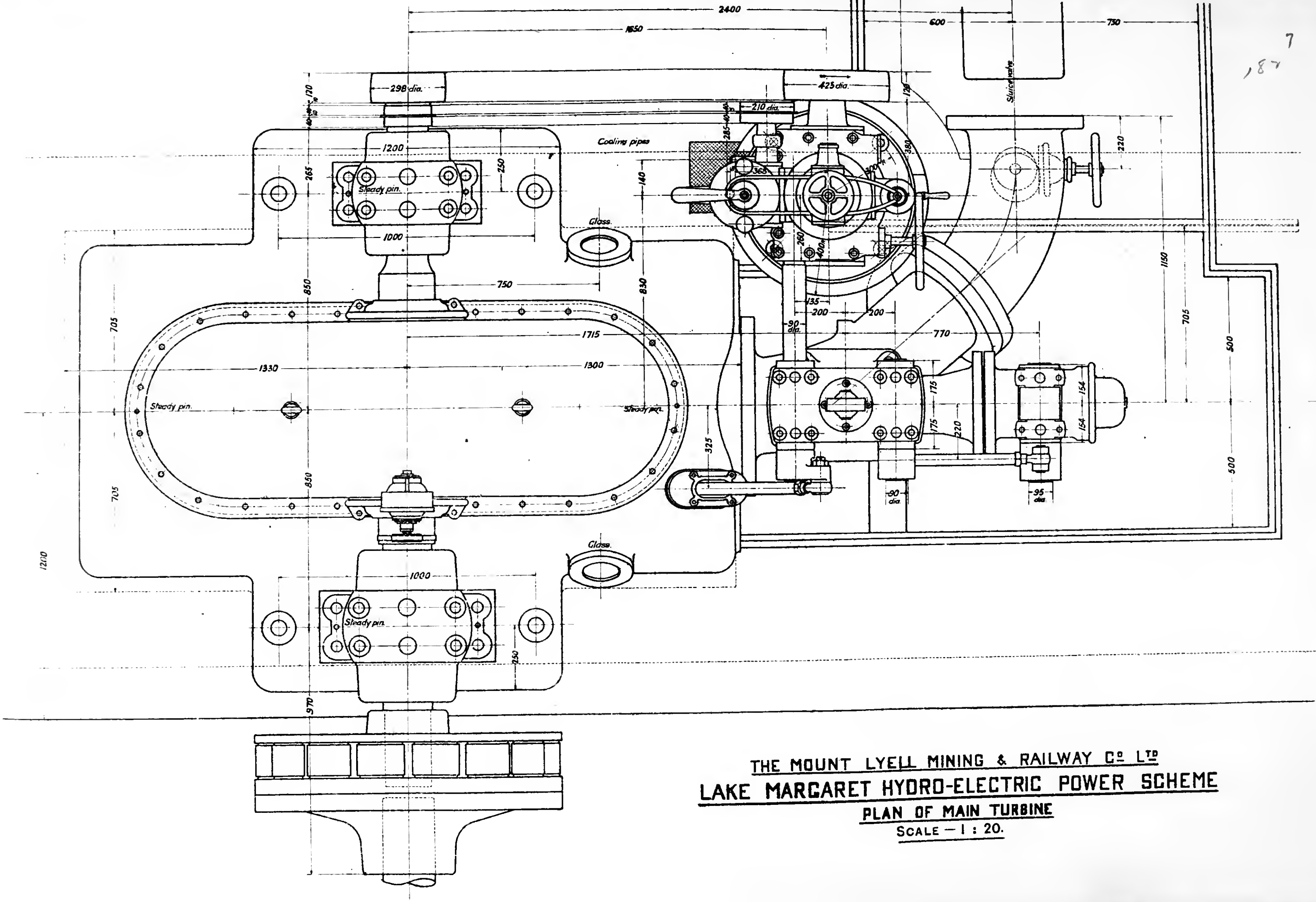
— Section C-D —



THE MOUNT LYELL MINING & RAILWAY Cº Lᵀᴰ
LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME
ARRANGEMENT OF POWER STATION.
SCALE — 1 : 100.


THE MOUNT LYELL MINING & RAILWAY Cº LTD
LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME
SECTION OF MAIN TURBINE
SCALE — 1 : 20.
BOVING AND Cº LTD. LONDON.
VERKSTADEN KRISTINEHAMN SWEDEN.
A.D. 1913.
Nº
21 BUCKETS.
1400 dia.
Tachometre
Pressure Gauge
Open
Shut
Stroke-90
Spring


215
To be filled in with concrete at erection
510
400
700
Ladder
950

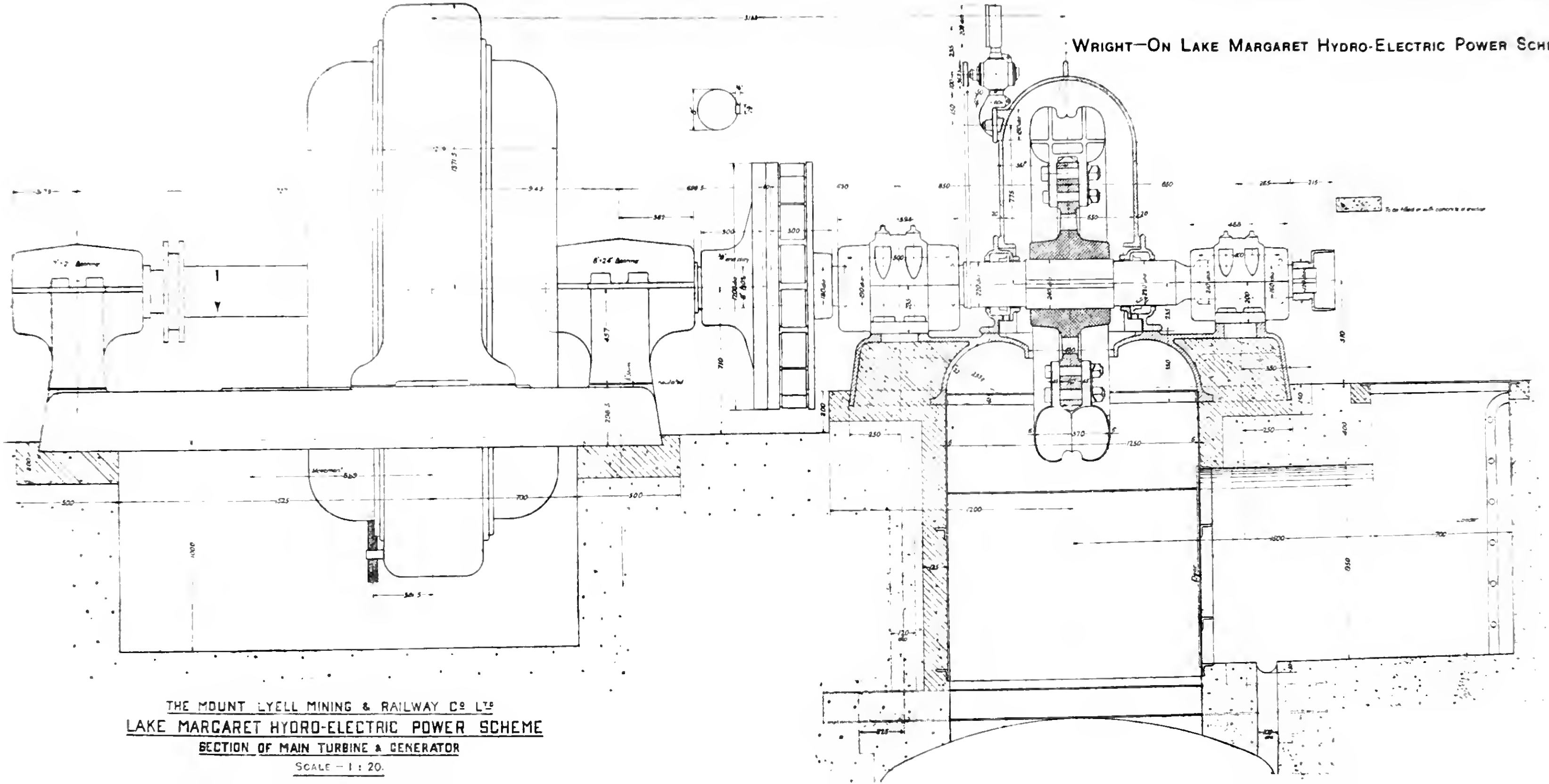


THE MOUNT LYELL MINING & RAILWAY C^O L^TD

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME

PLAN OF MAIN TURBINE

SCALE — 1 : 20.

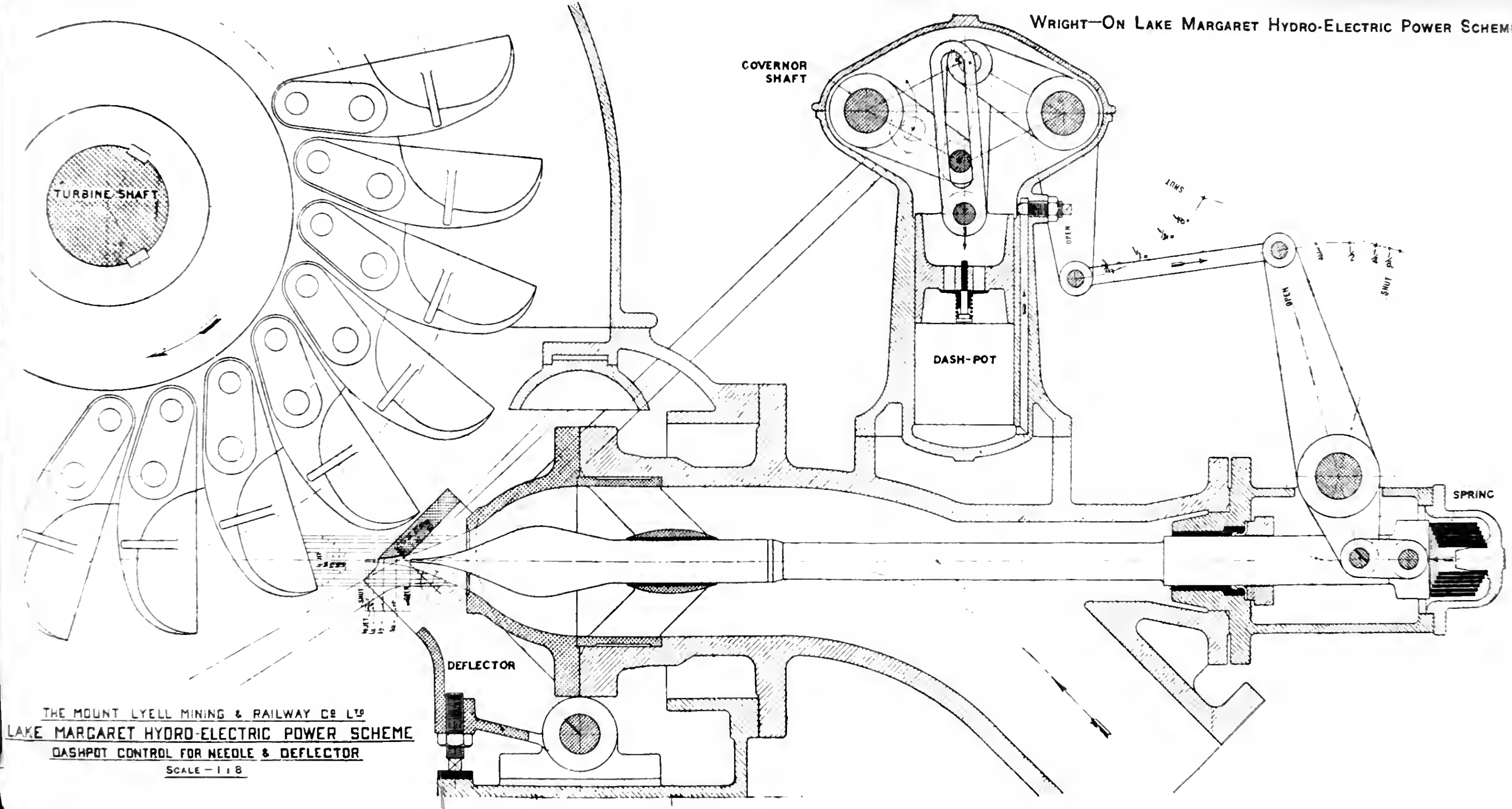


THE MOUNT LYELL MINING & RAILWAY C° L^TD

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME

SECTION OF MAIN TURBINE & GENERATOR

SCALE – 1 : 20.



THE MOUNT LYELL MINING & RAILWAY Co. Ltd

LAKE MARGARET HYDRO-ELECTRIC POWER SCHEME

DASHPOT CONTROL FOR NEEDLE & DEFLECTOR

SCALE – 1 : 8

NOTES ON THE TREATMENT OF STANNITE ORE AT ZEEHAN, TAS.

BY J. H. LEVINGS.

ALTHOUGH the mineral Stannite (Cu_2S, FeS, SnS_2, also known as "bell metal" ore, tin pyrites, etc.) is found in various parts of the world, in only three localities has development proved it to exist in sufficient quantity to warrant the erection of treatment plants. Two of those mines are in New South Wales and one is in Tasmania. The writer will deal with the latter—the Oonah mine, Zeehan, Tasmania, it being the most important and extensive occurrence so far worked.

The Oonah mine was first worked by a Tasmanian company for silver-lead, and was fairly remunerative until the zone of secondary enrichment was passed through, when the lodes became too poor for further exploitation. During the time that the mine was thus operated the stannite lode was discovered. It occurs in a graphitic slate, which, in common with the slates and spilites of the Zeehan field, is probably underlain by the stanniferous granite which outcrops two miles to the west. The Oonah stannite lode is older than the galena lodes of the district, being faulted by one of the latter. Near the surface the lode was very rich. Shipments were sent out assaying 150 oz. of silver per ton, 20 % copper, and 18 % tin. An apparently homogeneous sample of stannite, analyzed by the writer for the Tasmanian Geological Survey, gave the following results:—

	%			%
Gold ..	trace	(7 gr. to the ton)	*Brought Forward*	65.658
Silver ..	0.298		Zinc	0.475
Tin ..	23.57	(as sulphide)	Antimony ..	0.505
,, ..	0.64	(as oxide)	Arsenic ..	trace
Copper ..	26.77		Sulphur ..	30.1
Iron ..	12.11		Silica ..	1.40
Bismuth	2.27		Oxygen ..	0.14
Forward	65.658			98.278

[This paper is separately bound, and may be, if so desired, detached complete from this number.]

The lode was patchy, and, as depth was attained, changed to a siliceous iron pyrite, carrying a percentage of stannite. Still, some thousands of tons of hand-picked ore were shipped by the old company, which would average 11 % copper, 9.5 % tin, and 80 oz. of silver per ton. In some shipments a small allowance was paid for the tin, but mostly the ore was sold for its silver-copper values, with a returning charge approximating £5 per ton.

Eventually the mine became the property of an English company operating under the name of the Oonah Mines Limited.

This company further tested the stannite lode by crosscuts driven from the old galena levels. In the upper levels the lode was of the usual patchy nature, but in the bottom level a big, strong formation of primary sulphides came in, consisting of iron pyrite, stannite, and arsenical pyrite. This ore-shoot was 150 ft. below the zone of secondary enrichment, and gave promise to carry its values in depth. Bulk samples assayed 14 oz. of silver per ton, 3.5 % copper, 2.0 % tin, and 2.0 % arsenic. Had the lode been developed by deeper levels it would have been geologically interesting to note its characteristics as the granite was approached.

Unfortunately, the Oonah Mines Company did not derive much benefit from the big ore-shoot, as the latter did not go up more than 30 ft. in the stopes. Exhaustion of the ore reserves developed, coupled with the disappointing result of the treatment process, led to the cessation of mining. This was the more to be regretted, as, so promising were the prospects, that Government assistance for sinking could have been obtained upon application.

The three reduction plants erected in different localities to treat stannite ores first used reverberatory furnaces for their separations. The Conrad mine, in New South Wales, depended on an iron precipitation of the tin, while the Oonah and Tolwong, N.S.W. (the latter copying the Oonah), used a carbon precipitation. As the reverberatory processes were unprofitable, and gave no indications of ultimate success, it will not be necessary to go into details concerning them.

At the Oonah plant, however, a water-jacket blast furnace had been built to smelt the reverberatory slags. This furnace, being

intended only for the teatment of slag, had but one tier of water-jackets, the rest of the shaft being built of brick. As reduction was required, the distance from the tuyeres to the feed floor was made 25 ft. When the non-success of the reverberatory process was manifest, pyrite, or partial pyrite smelting—the practice being just on the border line—was undertaken in this furnace. It was successful from the start, a charge concentration of 15 into 1 being obtained, with a coke consumption of 4 %. Having such a high shaft, it was possible to make many experiments in the height of the charge column. Thirteen ft. was found to give the best results with the blast obtainable.

Also, the high shaft was found to be of great service when the furnace became crusted on top, and the blast was hissing through "blow-holes," with a scanty fall of high-grade matte below, signs that presage the early extinction of the ordinary pyrite furnace. Then the coke on the charge was increased to 8 or 10 %, and the furnace rapidly filled up 3 or 4 ft. In the course of some hours the scaffold would smelt out and the charge fall bodily 2 or 3 ft. The furnace would now be quite free and in good condition. By promptly cutting the coke again to 4 %, and lowering the stock line, the matte would average normal right through the operation. By this procedure a campaign might have been continued indefinitely, if it had not been for the brick-work of the walls above jackets burning out. Every stoppage was occasioned by this cause, the longest campaign being three months.

In the pyrite smelting about 50 % of the tin was volatilized, the remainder mostly going into the slag, and the balance into the matte. The silver recovery was fairly good.

It became imperative, however, to save the tin. In smelting the reverberatory slags for this metal it was found necessary to make a low-grade matte in order to keep the crucible of the blast furnace open, the matte, however, not interfering with the tin-recovery beyond retaining the tin, as sulphide, which had been present as pyrite before smelting. Nevertheless, a scheme was outlined to blast-roast the ore, with 15 % added iron-manganese

oxides, down to 8 or 9 % sulphur, and then smelt in the blast furnace under severely reducing conditions. This was put into practice. The products were a 20 % copper matte, containing 42 oz. silver per ton, and a copper-tin alloy, containing silver 660 oz. per ton, copper 42 %, tin 40 %, the balance being made up of lead (from the ironstone flux), bismuth, antimony, arsenic, and a little iron. The tin and copper approached the proportions in stannite—$Sn\ Cu_2$. The alloy was pure tin—white in colour, notwithstanding its high copper content. The iron contents rapidly oxidized, giving the alloy a rusty appearance on the exterior.

For this alloy, with a tin-copper-silver value of £200 per ton, no purchaser could be found in Australia. Afterwards, the writer believes, it was sold in England for £80 per ton.

On the termination of the metal-smelting campaign the furnace was put back on pyrite smelting, but the ore reserves were soon exhausted. The owners did not feel inclined to furnish the capital to put in the necessary new machinery to sink the shaft and open up the mine, so that the property was finally sold, but nothing further attempted.

The failure of the Oonah Mines Limited Company to profitably treat the stannite may be ascribed to two causes—firstly, insufficient capital for the undertaking; secondly, the failure of the reverberatory process which was first installed, and used up the funds, to the detriment of the subsequent work.

Though not successful, the Oonah Company in the later operations was undoubtedly on the right course to become so.

From the experience gained the writer will now propound a treatment process which he mapped out, but had no opportunity to put into practice. It may be of interest, and perhaps of service, to some metallurgist who may have a stannite proposition to handle.

In the blast-roasting of the stannite no volatilization loss of tin could be detected, although the pots were blown as briskly as possible. In the pyrite smelting quantities of a greyish sublimate readily formed on the sides of the cupola, and, had the

furnace been provided with dust-chambers, a quantitative amount could have been collected. On investigation it proved to be nearly pure tin oxide. This suggested a method for separating the tin from the silver and copper, and laboratory experiments carried out on these lines were satisfactory. Reasoning from the collected data, the following scheme was outlined :—

(1) A preliminary roast in mechanical furnaces, in part, or wholly, of the stannite ore, to be followed by blast roasting.

(2) Smelting of the roasted product under reducing conditions, producing tin-copper alloy, and sufficient 30 % matte for the next operation.

(3) Blowing the alloy and matte in the usual copper converter into silver-copper bullion and volatilized tin oxide, the latter to be collected in flue-chambers, which should be sufficient without a bag-house.

(4) Reduction of the tin oxide in a reverberatory or disposal to a tin smelter.

For the sake of comparison for costs, it may be taken that the Oonah mines were working on the above indicated process, and that the lode was of the same value and proportions as where exposed in the bottom level. All figures given are considered conservative.

Ore Assay.—3.5 %, 2 % tin, 14 oz. silver.

Gross Value, at current prices.—£9 3s. 6d. per ton.

Recoveries.—Copper 85 %, tin 55 %, silver 80 %.

Net value, £6 7s. 4d.

Mining (all costs)	£1 2s. 6d. per ton
Roasting, smelting, converting, and realization (all costs)	£2 5s. 0d.
	£3 7s. 6d. per ton

Leaving a surplus of £2 19s. 10d. per ton.

There is always a tin loss of 0.3 % or 0.4 % in the slag. The converter slag would show a higher content, but, of course, it would be sent back to the blast furnace.

Before closing these notes it may be as well to give, in brief, the assay methods used for the silver, copper, and tin in the stannite :—

Silver.—The usual scorification assay for copper ores gives reliable results when run with a good fire and a little extra test lead and borax.

Copper.—If the usual method of attacking with nitric acid, then adding sulphuric acid and taking to fumes, be adopted, the result will be about 20 % low.

On diluting and boiling the sulphuric solution the tin is thrown down as a flocky gelatinous precipitate of metastannic acid, which obstinately retains the copper, despite repeated washings.

A better method is :—Attack with nitric acid and evaporate until pasty, dilute, boil, and decant through a filter paper. Add a little nitric acid and hot water, stir the precipitate thoroughly, boil, let settle, and decant through the same paper, wash precipitate on to the filter, and wash several times with hot water. The copper is now in the filtrate, to which add sulphuric acid, take to fumes, and proceed as usual. The precipitate may be ignited, and used for the tin determination.

Tin. — Fuse over a burner in a nickel crucible with sodium peroxide. Cool, dissolve out with water into a flask, make acid with strong hydrochloric acid, then add 50 cc. in excess. Boil with about 25 grm. of nickel foil for 40 minutes. Add a piece of marble, cool rapidly, add starch solution, and titrate with standard iodine solution.

MINING EDUCATION IN AUSTRALASIA.

BY D. B. WATERS.

Discussion.

MR. WM. POOLE wrote:—Professor Waters had drawn attention to the important subject of Mining Education in Australasia. That matter had not in the past, in Australasia, received the attention and close scrutiny of the leaders of the mineral industry that its importance deserved. The writer was of the opinion that the diversity of diplomas issued at various institutions throughout Australasia was of minor importance, the essential consideration being that efficient technical instruction should be provided, suited to the needs of technical experts and the various grades and classes of men controlling mining and metallurgical operations. In order to accomplish that, it was necessary to have various grades of institutions, provided with suitable equipment and carefully-arranged courses of instruction, and controlled in such a manner as to obtain the most efficient results. Professor Waters' classification of Schools of Mines into three groups was a satisfactory one. His information regarding Sydney University was not accurate. There was a degree of Bachelor of Engineering (B.E.) in mining engineering and metallurgy, not a separate degree in each division. The granting of the Master of Engineering (M.E.)—"Civil," "Mechanical and Electrical," also "Mining and Metallurgy"—is, in addition to those quoted by Professor Waters, encumbered with the following regulations, viz.:—The examination must be held at the Sydney University, and nowhere else (this was common to all Sydney degree examinations); the candidate must have obtained honours at his Bachelor degree, or subsequent examination, a condition not required for any other degree in Australasia, not even at Sydney; twelve

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months' notice of intention to sit for examination (three months would be ample notice). These extraordinary regulations in practice made the obtaining of the M.E. degree almost unobtainable; no one had ever presented himself in the Mining–Metallurgical division. There were, without doubt, too many Schools of Mines, especially in Victoria, where many of them were merely such in name. On the other hand, New South Wales was singularly lacking in efficient schools in its metal and coal-mining and treatment centres. Broken Hill, Cobar, Newcastle, Maitland, Lithgow, and Wollongong might have schools which, for efficiency in "mining" instruction, should be comparable in efficiency with those at Charters Towers, Kalgoorlie, Ballarat, and Bendigo. Some of the above-mentioned New South Wales centres had technical colleges in which the "mining" side was weak and disappointing. The instruction at each centre should be carefully drawn up to suit the requirements of that district, and not a standard curriculum for all and specially suited to none. The control of technical teaching institutions (including Schools of Mines) varied very much in Australasia, but may be divided into two main groups—

(*a*) Free from State administration or control.

(*b*) Partially or wholly under State control.

(*a*) These included the technical branches of the various Universities and the Adelaide School of Mines. These institutions were reasonably similar in their control, received directly or indirectly financial assistance from their respective States, their governing bodies usually contained State representatives, but the State did not exercise, through any of its departments, any control over their internal administration, curricula, examinations, degrees, or diplomas. (*b*) Here was found extraordinary diversity of types. The Schools of Mines at Charters Towers and Kalgoorlie came directly under the control of the Mines Department of their respective States. Those schools were under the supervision of directors, each of whom was responsible to the Minister and Under Secretary for Mines for the administration, discipline, standard and scope of work, and teaching efficiency. The examining body

was composed of the lecturers and outsiders acting conjointly. In Victoria the Schools of Mines had local governing councils subsidized by the State, the teaching staffs having little or no voice in the administration. The syllabus of subjects was set by the Department of Education, usually acting on the advice of special committees composed of instructors, examiners, and outside experts. The Department conducted the examination and issued certificates. Several of the older and larger institutions, such as the Schools of Mines at Ballarat and Bendigo, drew up their own diploma courses and issued diplomas, but the Department conducted the examination of individual subjects. The individuality of these old institutions was thus maintained. In New South Wales the Sydney Technical College and its numerous branches was a branch of the Education Department, which controlled its courses, syllabus of subjects, examinations and certificates. The teaching staff had no voice in those matters, but may be asked for its advice. That institution gave, *inter alia*, instruction in mining and metallurgical subjects, but in the past, despite its unique opportunities, it had lamentably failed to provide adequate and efficient instructions in its mining centres. This extensive institution was now being reorganized. In Queensland the various technical colleges may undertake mining and metallurgical instruction. The local administration and control of examinations and control of syllabi was somewhat similar to that of Victoria. The amount of work done was small. The system adopted at Charters Towers and Kalgoorlie was more suited to a single institution than a group. It gave excellent results, in the hands of capable men, but would be disastrous under incompetent control. The councils at provincial schools were usually too large, and were mostly composed of men whose only recommendation was their willingness to act. Those large councils should be replaced by small bodies of about five to seven specially selected members, of which the Principal should be one. The Principal should be one in reality, responsible for the efficiency of work and discipline of staff and students. It was worthy of note that the most efficient institutions were those that were free from Depart-

ment of Education control, or those—such as the Ballarat School of Mines—which had attained a high standing prior to such interference. That, at first sight, was surprising, until it was remembered that the chief experience and function of Departments of Education, especially in the past, had been primary education, and, despite their grandiose pretensions, they really understood very little of other branches, but especially of technical education. The technical branch of every Department of Education should be freed from the primary school influence, and placed under the responsible control of technical men of wide training and experience. The curriculum for a mining or metallurgy, or combined mining-metallurgy, degree should be most carefully drawn up, as to the subjects included, the synopsis of each subject, and the general balance of the course. Students usually had but a limited time (three or four years) at their disposal for their technical education, which should therefore take the fullest advantage of that time. The instruction at the Universities should be more seriously treated as training for a profession than as academical culture of an applied science nature. A thoroughly sound professional training should be compatible with the undoubted advantage of the refinement of a University education. Professors and lecturers in science subjects were still seen at Universities who openly or in practice abhored set work of professedly utilitarian nature. Curricula were frequently sadly out of balance, owing in part to professors of pure science subjects obtaining an undue allotment of time for their work, to the detriment of "professional" subjects, the lectures of some of which were given by "outside" lecturers. Unfortunately, too many of the lecturers in engineering and other applied science subjects at the Universities and other technical institutions were academical men with little or no "works" experience in their subject, and who, while their scientific knowledge may be sound, frequently gave prominence to unimportant matters and neglected others of professional value. University faculties, boards of study, and syllabus committees should contain scientifically trained professional men of wide practical experience, who personally knew

what was essential, or desirable, in the training of young mining engineers or metallurgists, to enable them to efficiently carry out their duties in conducting working operations. The influence of such men may be seen in the radical modifications and extensions made a few years ago in the curricula of the Royal School of Mines. It was very desirable that there should be separate degree or diploma courses in mining and in metallurgy, rather than a combined course, as the latter became too general and scrappy in the treatment of the main professional subjects. The mining engineer should have a general elementary knowledge of metallurgy, and the metallurgist should have a general elementary knowledge of mining operations, so that each should intelligently understand the complementary portion of the industry. Separate curricula enabled the essentially professional subjects to be treated to a much-to-be-desired greater fulness than was in vogue in University combined degree courses. The following subjects, which were either not at all or imperfectly treated, should be carefully studied in University and University types of Schools of Mines. General engineering, but especially the electrical engineering side of the curricula, was usually weak; the increasing importance of electrical machinery and appliances did not receive the consideration that it warranted. The writer did not advocate that all mining engineers and metallurgists should be highly trained electrical engineers, but they should be conversant with the nature and application of electrical-power machinery and appliances. A similar knowledge of steam power had long been necessary. Economic or mining geology should in many courses be taken in a much more comprehensive manner, and in greater detail. It should include all commercial minerals as well as the more common ores. The term mineral was here used in its broad commercial sense. Mining engineers and metallurgists, much more than civil or mechanical engineers, frequently had to supervise operations of a highly commercial nature. The new curricula of the Royal School of Mines introduced lectures and demonstrations in business transactions. Lectures were usually given on the mining laws and regulations of the State in which the

School was situated; but, owing to the peripatetic nature of the mining profession, it was highly desirable to study the terms of tenure, regulations, etc., in other States. Lectures should also be given in law of contracts, industrial laws, inspection of machinery regulations, etc. It was not advocated that a mining engineer or metallurgist should be his own lawyer, but he should have a sound knowledge of the legal requirements of his various operations. The mechanical handling of materials was essential in all mining and metallurgical work. A detailed course of lectures should therefore be provided. Full courses of lectures, and, where practicable, demonstrations, should be given in sanitation and tropical hygiene, mine reporting and valuation, the arrangement and design of mining and metallurgical plants. An essential feature in all first-class Schools of Mines should be the possession of engineering, ore-dressing, and metallurgical laboratories. The primary objective of those should be teaching purposes. Students should work singly or in groups in experimentally working out problems involving fundamental principles and processes. Original research was very valuable for both staff and post-graduate students, but, where only limited funds were available, special plant and apparatus for research should not be installed to the curtailment of teaching plant. Students should receive very full practical instruction in work they were likely to be called upon to undertake in their early outside employment —viz., sampling, assaying, surveying, and draughting. They should be so trained as to be of material value to their first employers. It was an unfortunate fact that too many graduates who, despite their sound scientific training, were for a considerable time practically helpless because they were ignorant of the minor usages and routine of their work, placing them in an unnecessarily humiliating position, and casting ridicule upon their Alma Mater. It was both unnecessary and unwise to insist on long periods of practical experience as one of the conditions of obtaining a degree or diploma. If several years of practical experience were required, it would act as a severe deterrent to obtaining a School of Mines' diploma, unless such a diploma were necessary

by law. Professor Waters had pointed out several methods by which students might obtain their necessary practical experience—viz., during scattered vacations and after the close of their course, or by the sandwich system. There was another method—viz., concurrently with their studies. That obviously could only be done at Schools of Mines in centres where such opportunity was available, and was of more application to mining than to metallurgical experience. At the Charters Towers School of Mines the teaching work was so arranged that any instruction given in the forenoon might be duplicated in the evening, thus enabling students to attend lectures, etc., whether they were working on day, afternoon, or night shift. Working students may take as few subjects per year as they wished, and extend the graduate course to any length of time they may require, usually four to six years. About two-thirds of the diplomas had been granted to students who had worked on the concurrent method. Educationally, the results had been excellent, and many such graduates now held very responsible positions. That method also enabled a student to make a living and pay for his education. It also brought forward young men of grit, determination, and ability. Most technical and mining schools now recognized each other's certificates, so that a student may go from a less to a better-equipped school to continue his diploma work, obtaining recognition for previous work. Most of the Universities also granted partial recognition for work done at approved mining or technical schools, provided the student matriculated. The State Mines Departments of Queensland and Victoria recognized the degrees or diplomas in mining of Schools of Mines, provided the range and standard of work covered by the degree or diploma was equal to that required for the mine managers' certificates. If the diploma did not include an examination in the mining regulations of the State in which application was made, it was necessary for the applicant to pass such an examination before the certificate was granted. The applicant must produce a certificate in "First Aid to the Injured," issued by some recognized body, also certificates to show that he had the necessary practical mining

experience. Neither of those two States required applicants to have obtained at the granting of the School of Mines' diploma the amount of practical experience required for the mine manager's certificate. It had long been apparent to outside observers that the Mines Department of New Zealand had adopted a hostile attitude towards School of Mines' diplomas. The action of the Mines Departments in Queensland and Victoria showed that the gulf could be satisfactorily bridged. In New South Wales the Department was not hostile, but the wording of the Mining Act did not permit of recognition of diplomas. It was quite common at Schools of Mines on mining fields to find students who (*a*) did the whole or most of their practical experience required for mine manager's certificate after they had completed their academical work, or (*b*) had done the necessary amount of practical work before commencing their academical work. It was very desirable that the qualifications for the various State mining certificates should be standardized, and that the certificates issued by one State should be recognized by the other States. Such proposed line of action would not be novel, as it was many years since the qualifications were standardized and the examinations were federated for licensed surveyors in Australasia.

PROCEEDINGS

OF THE

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No. 21.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

1916.

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Australasian Institute of Mining Engineers.

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Head Office:

57-59 SWANSTON STREET, MELBOURNE, VICTORIA.

Institute Matters.

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MINUTES OF MEETINGS.

OF THE INSTITUTE.

ANNUAL MEETING.

MELBOURNE, 28TH JANUARY, 1916.

IN THE INSTITUTE ROOMS, 57-59 SWANSTON STREET, AT 1 P.M.

Mr. H. Herman occupied the chair.

The Secretary read the notice convening the meeting.

Minutes of the First Ordinary Meeting, 1915, were confirmed.

The report of the Council and Balance Sheet for 1915, were read and adopted.

It was resolved that, in connection with the proposed Federal Institute for Research the Council be recommended to keep in view the desirability of obtaining representation on that Institute.

The result of the ballot for the election of officers to fill the vacancies was reported as follows :—

President - -	Robt. C. Sticht, Tas.
Vice-President -	W. E. Wainwright, N.S.W.
Council - -	R. S. Black, W.A.
	G. D. Delprat, N.S.W.
	Geo. H. Broome, Vic.
	Richard Hamilton, W.A.
	A. S. Kenyon, Vic.
	George C. Klug, W.A.
	George Weir, N.S.W.

The meeting then closed.

Report of the Council for 1915.

TO THE MEMBERS.

The Council has pleasure in submitting the following report of the Proceedings of the Institute for the year 1915.

The number of Members on the Register is now 678, made up as follows :—Honorary Member, 1 ; Fellow, 1 ; Life-Members, 5 ; Members, 426 ; Associate Members, 173 ; Students, 71 ; Subscribers, 2.

Twenty-eight Members resigned during the year and Forty-one were removed from the register for non-payment of subscriptions.

The Council records with regret the loss, by death, of four of its Members, viz. :—R. S. Anderson, David M. Davis, H. J. Laidler, V. C. Stuckey. The names of three Members killed in action at the front are reported under another heading.

During the year Professor D. B. Waters was appointed to fill a casual vacancy accurring in the office of Vice-President.

Three Meetings of the Institute were held—The Annual Meeting on the 29th January, a Special General Meeting on the same date, and the First Ordinary Meeting, 1915, from 5th March to 14th March.

The First Ordinary Meeting, 1915, included visits to the following Mining Centres in the North West of Tasmania:—Queenstown (including Mount Lyell), Zeehan (including North Dundas), Rosebery and Waratah (including Mount Bischoff). Necessarily, most of the time was occupied inspecting the Mount Lyell and North Lyell Mines, where Mr. R. C. Sticht (President) and his staff took charge of the party. Mr. James B. Scott of the Renison Bell Mine, Mr. C. Moxon of the Hercules Mine, Mr. G. Barker of the Tasmanian Copper and Primrose Mines and Mr. J. D. Millen of the Mount Bischoff Co. did the honors at their respective mines. A full report of the Meeting appeared in the Proceedings, No. 17 and those Members who were fortunate enough to accompany the party took full advantage of the opportunity afforded for examining one of the most interesting mining fields in the world. To the management and members of the staffs of the various mines, the Council desires to express its thanks and appreciation for the facilities afforded and the many kindnesses shown to the Members during the visit.

A recommendation was made at the Mount Lyell Meeting that the First Ordinary Meeting, 1916, be held in the South Island of New Zealand. Owing, however, to the abnormal conditions existing on account of the war the possibility of even a fair attendance at such a meeting was considered by the Council to be remote and the suggestion was abandoned.

Publications issued during the year were:—Proceedings (new series), No. 17, 18 and 19, also a complete index, issued as a Supplement to No. 18, of all papers issued by the Institute from its inception to the end of 1914. Members have been notified that they may, if they so desire, have their four quarterly Proceedings bound in dark cloth, lettered in gold on back, at a cost of 5s. 6d. plus postage.

During the year Mr. J. M. Bridge was appointed Local Secretary to the Broken Hill Branch.

A copy of the draft of the Memorandum and Articles of Association as suggested by the Broken Hill Branch, was published in No. 19 of Proceedings and Members were invited

to forward expressions of opinion thereon. Very few Members have responded to this invitation. The difficulties of incorporation pointed out in Proceedings No. 19 are emphasised under present war conditions, when it would be hopeless to expect legislators to take effective interest in the matter. The project will be revived at the first favourable opportunity.

The following Institutions have been added to the list of Societies, etc., which received the Publications of the Institute by way of exchange :—Silby College of Mechanical Engineering, Cornell University; Sheffield Scientific School, Yale University; and the Northern Engineering Institute of New South Wales.

Mr. W. G. Langford of the Melbourne University was awarded the "A. T. Danks Prize, 1915" for the best paper descriptive of the visit to Queenstown, Zeehan and Waratah in March.

Owing to the growth in numbers of visitors attending the First Ordinary Meeting of the Institute it has been decided that such meetings be confined to persons directly interested in mining and allied sciences. It has also been decided that the proceedings of such meetings be restricted to technical and mining business only.

Steps have been initiated to secure direct Institute representation on boards and faculties controlling mining education.

During the year a bill to provide for the registration of architects was framed by the Royal Victorian Institute of Architects in Melbourne. The bill received the attention of the Council and, as it was considered that the rights of Members of the Institute would be prejudiced by the bill as drafted, steps were taken to safeguard their interests.

At the invitation of the Munitions Committee it was resolved that the Institute become a sub-committee thereof.

Two Councillors (Herman and Skeats) were Members of the Interstate Conference, called by the Minister of Defence to advise on the formation of a Miners' Corps for war service. After the corps had been decided on these gentlemen also took part in the work of equipment, organization and selection of men and officers. The formation of the corps, in which the Institute is well represented, should be a source of pride to all connected with mining.

On the recommendation of Members at the First Ordinary Meeting the Council donated the sum of £50 to the Belgium Relief Fund.

The Council takes pride in recording the names of Members of the Institute who have answered the Empire's Call to Arms. They include :—

Aarons, J. Boyd
Alexander, Hubert
Anderson, E. S.
Anderson, W. T.
Avery, Wm. P.
Bayly, Colin (Killed in Action)
Bell, W. G.
Bennett, V. C.
Berriman, R. V.
Best, G. H. T.
Bray, F. P.
Blunden, L. E.
Brown, H. Wheeler
Campbell, A. G.
Casey, R. G.
Connor, J. L. (Killed in Action)
Coulter, L. J.
DeGrut, L. J.
Dempster, G. C.
Donaldson, R. J.
Fraser, D. G.
Fraser, E. H.
Gabriel, E. E.
Goode, K. B.
Herbertson, R. C.
Holder, E. M.
Hooper, F H.
Hunter, Stanley B. (Member of Council)
Irvine, C. L.
Jowett, H. C.
Kelly, M. B. H.
Key, J. F.
Lewis, K. B.
Marks, D. G.
Mawdsley, W. H.
Mickle, K. A.
Moore, R. I.
Mulligan, E. N.
MacDonnell, W. H. A.
M'Bryde, J.
Nicholas, Frank H.
Pearson, H. F.
Roberts, R. A. J.
Rose, W. J.
Sayer, W. T.
Seale, H. V.
Smith, E.
Smith, R. S.
Southon, R. D.
Sweet, O. G.
Tandy, A. E.
Townsend, H.
Tucker, V.
Turner, W. A.
Wallmann, H. P.
Waters, Professor D. B. (Vice-President)
Watts, R. T. (Killed in Action)

A number of other Members, it is believed, have enlisted but have not reported the fact to the head office and the Council

would be glad to receive names and particulars. It has been decided that all Members who have been accepted for Active Service be exempt from payment of Subscription during their period of Service. It will be seen that the roll of honor includes the names of three Members who have given their lives for the Empire, viz. :—Colin Bayly, killed in action at Flanders ; Lieut. J. L. Connor and Raymond T. Watts, killed in action at Gallipoli. To the relatives of these men the Council tenders its sincere sympathy.

As was to be expected the Institute has suffered somewhat owing to the war, but its financial position is nevertheless sound. Commonwealth Stock to the amount of £250 was purchased during the year and the fixed deposit in the Bank of Australasia remains at £200. It will, the Council feels assured, be the earnest endeavour of each Member to assist in maintaining the strength of the Institute during the present troublous times, and this can be done in several ways, viz. :—(1) by introducing new Members, (2) by preparing papers for reading and discussions, (3) by prompt payment of Subscriptions.

AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

Balance Sheet, 31st December, 1915.

LIABILITIES.

	£	s.	d.	£	s.	d.
Sundry Creditors				£188	16	6
Subscriptions paid in advance				22	14	6
Accumulated Fund—						
Balance at 31st Dec., 1914	£604	11	6			
Less Deficiency transferred from Revenue A/c	65	5	3			
				539	6	3
				£750	17	3

ASSETS.

	£	s.	d.	£	s.	d.
Bank Balance in Bank of Australasia	£67	18	11			
Cash in State Savings Bank of Victoria	9	11	10			
				£ 77	10	9
Fixed Deposit in Bank of Australasia ...				200	0	0
Federal Government War Loan Debentures				250	0	0
Outstanding Subscriptions, 1914	21	0	0			
,, ,, 1915	129	10	0			
				150	10	0
Sundry Debtors for Advts., &c.	36	3	4			
Suspense Account — Cash in hands of New Zealand Branch	2	16	3			
				38	19	7
Office Furniture, Melbourne and Broken Hill Branch ...	38	1	6			
Less Depreciation	4	4	7			
				33	16	11
				£750	17	3

Melbourne, 22nd January, 1916. Audited and found correct. B. H. Oxlade, A.C.P.A., Auditor.

Statement of Receipts and Expenditure for the Year ended 31st December, 1915.

RECEIPTS.

	£	s.	d.	£	s.	d.
To Bank of Australasia, Balance, 1914 ...	£19	7	7			
State Savings Bank of Victoria, Balance, 1914	36	4	8			
Cash on Hand, 1914 ...	15	15	6			
				£71	7	9
Members' Subs., 1912 ...	4	4	0			
,, ,, 1913 ...	10	11	6			
,, ,, 1914 ...	81	2	0			
,, ,, 1915 ...	843	3	3			
,, ,, 1916 ...	22	14	6			
				961	15	3
Advertisments				50	0	0
Sales of Publications				6	0	1
Interest				7	14	4
Entrance Fees				36	4	7
				£1133	2	0

EXPENDITURE.

	£	s.	d.	£	s.	d.
By Postages, Telegrams, &c.	£54	17	5			
Printing & Stationery ...	23	8	0			
Salaries	236	0	0			
Rent	60	0	0			
				£374	5	5
Sundries				26	9	0
Meeting Expenses				12	11	4
Branches				50	0	0
Proceedings				285	8	0
Commission				6	17	6
Donation—Belgium Relief Fund ...				50	0	0
Federal Government War Loan Debenture				250	0	0
Bank of Australasia, Balance at Credit ...	67	18	11			
State Savings Bank of Victoria, Balance at Credit	9	11	10			
				77	10	9
				£1133	2	0

Revenue Account for the Year ended 31st December, 1915.

Date	Particulars	£	s.	d.	£	s.	d.
1915.							
Dec. 31.	To Publications	£313	13	3			
	Meeting Expenses	12	11	4			
	Branches ...	50	0	0			
	Stationery & Printing ...	22	7	0			
	Postages ...	54	17	5			
	Salaries ...	257	0	0			
	Rent ...	48	0	0			
	Sundries ...	26	9	0			
	Commission	6	17	6			
	Donations—Belgium Relief Fund	50	0	0			
					£841	15	6
	Bad Debts—Subscriptions	331	10	3			
	Depreciation (Furniture)	4	4	7			
					335	14	10
					£1177	10	4

Date	Particulars	£	s.	d.
1915.				
Dec. 31.	By Subscriptions, 1915 ...	£1001	2	9
	Advertisements and Sales ...	53	10	1
	Interest	21	7	8
	Entrance Fees	36	4	7
		1112	5	1
	Deficiency for year transferred to Accumulated Fund ...	65	5	3
		£1177	10	4

Melbourne, 22*nd January*, 1916. Audited and found correct. B. H. Oxlade, A.C.P.A., *Auditor*.

OF THE COUNCIL.

(Abstract.)

FEBRUARY 7TH, 1916—1 P.M.

It was resolved that an Executive Committee be appointed for 1916, consisting of all members of the Council resident, or, for the time, being in Melbourne.

OF THE EXECUTIVE COMMITTEE.

(Abstract.)

FEBRUARY 7TH, 1916—1.15 P.M.

The Secretary's report was presented and accounts to the amount of £209, including printing Proceedings, £167, were passed for payment.

Nomination of Mr. G. D. Reid as an Associate Member and Mr. C. F. Duffield as a subscriber were approved. That of Mr. Godfrey Stevenson was deferred.

Sub-Committees for 1916 were appointed.

Estimates of receipts and expenditure for 1916 were submitted by the Finance Committee and adopted.

Papers by J. C. Coldham on "Organization and Equipment of a Mine-Rescue Station" and by A. M. Matheson on "Notes on Chemical Assay of Tin Ores" were referred to the Publication Committee.

A request that money collected by Mr. Peter Sutherland for the purchase of Band Instruments for the Mining Corps should be donated through the Institute was approved.

Resolved that in connection with the Federal Research Institute a special meeting of the Executive Committee be held on Monday next to consider the question of co-ordination of Universities and Technical Mining Schools with Mining and allied Sciences.

Other routine business was transacted.

FEBRUARY, 14TH, 1916—1 P.M. (SPECIAL MEETING).

The question of the proposed Federal Institute for Research was discussed. It was reported that several Mining Companies had agreed to subscribe towards bursaries for Students and to undertake to provide employment for a limited period afterwards.

It was agreed that Professor E. W. Skeats represent the Australasian Institute of Mining Engineers at a meeting of the donors which would shortly be held.

MARCH 3RD, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £62, including printing £22, were passed for payment.

A paper entitled "Scheelite-Gold Mines of Otago, N.Z." by C. W. Gudgeon was referred to the Publication Committee.

It was reported that Band Instruments had been formally presented to the Mining Corps by the Institute and that the balance of funds collected would be forwarded to the Mining Corps' Comforts Depot in Sydney.

A suggestion to assist families of members of the Mining Corps who were in very poor circumstances, was submitted, and it was decided that the Institute lend its assistance, but that it could not take charge of the work.

It was reported that objections had been taken on behalf of the Institute to the registration as a Union of an Association called the "The Australian Union of Architects, Engineers, Surveyors and allied Professions" and it was resolved that a Special Meeting of the Committee be convened to consider the matter.

Other routine business was transacted.

MARCH 8TH, 1916—1 P.M. (SPECIAL MEETING).

It was resolved that the Institute co-operate with other Institutes in opposing the registration as a Union, of an Association named "The Australian Union of Architects, Engineers, Surveyors and allied Professions as an organization of employees." Objection was to be made on account of the title only.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Application for admission to the Institute have been received from the following :—Member—Mr. CHARLES M. LYONS, Tavoy, Burmah. Associate Member—GEORGE DAVID REID, Oolda Wirra, S.A.

PUBLICATIONS.

Papers appearing in this number are :—

- Notes on Chemical Assay of Tin Ores. By A. M. Matheson.
- The Organization and equipment of a Mine-Rescue Station. By J. C. Coldham.
- The Scheelite-Gold Mines of Otago, New Zealand. By Cyril W. Gudgeon.

COMMONWEALTH INSTITUTE OF SCIENCE AND INDUSTRY.

A Committee has been formed to consider the question of the co-ordination of the Universities and Technical Mining Schools with mining and allied sciences. In this connection a scheme has been drawn up by a Committee representing a number of the principal mining companies for the encouragement of suitable candidates who intend adopting the professions of mining-engineering and metallurgy. The Institute will be represented on this Committee by Professor E. W. Skeats (Member of Council).

SUPPLEMENTARY LIST OF MEMBERS WHO HAVE ENLISTED FOR SERVICE AT THE FRONT.

In addition to the list appearing in the Annual Report the following Members have enlisted for active service :—

BANKS, CHARLES A.
BELL, DR. J. MACKINTOSH
CARROLL, H. H.
COWLES, RICHARD K.
EDWARDS, EDWARD
KNEEBONE, C. S.
MARRYATT, CYRIL B.
M'PADDEN, J. F.
MORRIS, A. C.
RUTHERFORD, A. R.
RUTHERFORD, T. W. L.
VEITCH, NEIL A.
WOODWARD, J. J.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

From 31st December, 1915, to 31st March, 1916.

Australian Mining Standard - -	weekly -	Melbourne
Australian Mining and Engineering Review	monthly -	Melbourne
Engineering and Mining Journal - -	weekly -	New York
Iron and Coal Trades Review - -	weekly -	London
Mining Journal - - - -	weekly -	London
Mining and Scientific Press - -	weekly -	San Francisco
The Colliery Engineer - - -	monthly -	Scranton, Pa.
Mining and Engineering World - -	weekly -	Chicago
Mining Magazine - - -	monthly -	London
Indian Engineering - - -	weekly -	Calcutta
Chemical News - - - -	weekly -	London
South African Engineering - -	monthly -	London
Journal of Industrial and Engineering Chemistry - - -	monthly -	Easton, Pa.
Society of Chemical Industry : Journal -	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal - -	monthly -	Johannesburg
Franklin Institute : Journal - -	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Metallurgical and Chemical Engineering -	monthly -	New York
Chamber of Mines of Victoria : Monthly Mining Report - -	-	Melbourne
Chamber of Mines of Western Australia : Journal - - - -	monthly -	Kalgoorlie
The West Australian Mining, Building and Engineering Journal - - -	weekly -	Perth
Queensland Department of Mines : Government Mining Journal - -	monthly -	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Production -	-	Johannesburg
Rhodesia Chamber of Mines : Report of Executive - - -	monthly -	Bulawayo
Department of Mines, New South Wales : Mineral Resources, No. 20 - -	-	Sydney
Northern Engineering Association of New South Wales : Transactions, Vol. VI. - -	-	Newcastle

Royal Society of New South Wales :	
Journal and Proceedings, Vol. XLIX., Part 2 -	Sydney
University of Adelaide : Calendar, 1916 - -	Adelaide
Royal Society of South Australia :	
Transactions, Vol. XXXIX. - -	Adelaide
School of Mines, Adelaide : Annual Report, 1915 -	Adelaide
Department of Mines, Western Australia :	
Annual Progress Report of Geological Survey, 1915	Perth
Royal Society of Tasmania :	
Papers and Proceedings, 1915 - -	Hobart
Geologists' Association :	
Proceedings, Vol. XXVI., Part 5 - -	London
Institution of Mining and Metallurgy :	
Bulletins, Nos. 134-136 ... - -	London
Institution of Chemistry of Great Britain and Ireland :	
Proceedings, Parts 1-4 - - -	London
Institution of Mining Engineers :	
Transactions. Vol. XLIX., Part 7 ; Vol. L., Parts 1 and 2 - - - -	London
Royal Dublin Society :	
Economic Proceedings, Vol. II, No. 10	
Scientific Proceedings, Vol. XIV., Nos. 24-41 -	Dublin
Société Des Ingenieurs Civils De France :	
Memoirs, Series 7, No. 11	
Bulletin, Nos. 9 and 10, 1915 - -	Paris
Societa Geologica Italiana :	
Bulletin, Vol. XXXIV., Part 3 - -	Rome
Geologisches Comite :	
Bulletin, Vol. XXXI., Nos. 9 and 10; Vol. XXXII., No. 1 - - - -	Petrograd
Mysore Geological Department : Records, Vol. XIV. -	Bangalore
Canadian Mining Institute : Transactions, Vol. XVIII.	Montreal
Department of Mines, Canada :	
Electro-thermic Smelting of Iron Ores in Sweden	
Bulletin, Nos. 20 and 21 - - -	Ottawa
American Institute of Mining Engineers :	
Bulletins, Nos. 108-110 - - -	New York
American Society of Civil Engineers :	
Transactions, Vol. LXXIX. - - -	New York

California State Mining Bureau:	
Various Reports by State Mineralogist -	Sacramento
Nova Scotian Institute of Science:	
Proceedings and Transactions, Vol. XIII., Parts 3 and 4; Vol. XIV., Part 1 - -	Halifax
Geological Survey of Alabama: Bulletin, No. 15 -	Alabama
Smithsonian Institute: Annual Report, 1914 -	Washington
United States Geological Survey:	
Water Supply Papers, Nos. 342, 356-358, 340 K, 375 B and 375 C	
Bulletins, Nos. 544, 566, 569, 587, 601 and 612	
Professional Papers, Nos. 95 B and 95 C	
Monograph, LIII. - - - -	Washington
University of Illinois: Bulletins, Vol. XII., No. 82 -	Urbana
El Museo Nacional:	
Annales Del Museo Nacional De Historia Natural, Tomo, XXVI. - - - -	Buenos Ayres

RECENT ARTICLES ON MINING MATTERS.

(31st December, 1915, to 31st March, 1916).

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly; mth., monthly.

(1) *The Australian Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk. 6d.

(5) *Mining and Engineering World*, Chicago, wk., 10c.

(6) *The Engineering and Mining Journal*, New York, wk., 15c.

(7) *The Colliery Engineer*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining and Scientific Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

(11) *Publications*, Department of Mines, Sydney, New South Wales.

(12) *Publications*, Department of Mines, Adelaide, South Australia.

(13) *Publications*, Department of Mines, Brisbane, Queensland.

(14) *Publications*, Department of Mines, Perth, Western Australia.

(15) *Publications*, Department of Mines, Hobart, Tasmania.

(16) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(17) *Publications*, Bureau of Mines, Toronto, Ontario.

(18) *Publications*, Geological Survey of India, Calcutta.

(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(24) *Quarterly Journal*, Geological Society, London.

(25) *Transactions*, Inst. Mining and Metallurgy, London, E.C.

(26) *Transactions*, Inst. Min. Eng., London.

(27) *Journal*, Canadian Mining Inst., Ottawa, Ontario.

(28) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(29) *Transactions*, Am. Inst. of Min. Eng., New York City.

(30) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(31) *Journal*, Franklin Inst., Philadelphia, Pa.

(32) *Australian Mining and Engineering Review*, Melbourne, Vic., mth., 6d.

(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins*, Société des Ingénieurs Civils, Paris.

(35) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(36) *Publications*, Iron and Steel Institute, London.

(37) *Proceedings*, Inst. of Mech. Eng., London.

(38) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

(41) *Publications*, Department of Mines, Wellington, N.Z.

(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(44) *Proceedings*, Geologists' Association, London.

LIST OF ARTICLES.

MECHANICAL.

The Hoisting Rope Question. (6) Dec. 4, 1915.

Leyner *versus* Piston Drill. L. P. Gordon. (6) Dec. 11, 1915.

Design for Mine Car and Rolling Side-Dump Tipple. R. F. Smith. (6) Jan. 22, 1916.

Necessary Use and Effect of Gas Compressors on Natural Gas Field Operating Condition. S. S. Wyer. (29) Bull. No. 110.

New Electric Hoist of the North Butte Mining Co. F. Moeller. (29) Bull. No. 110.

METALLURGICAL.

Building the Tough–Oakes Mill. John A. Baker. (6) Serial commenced Nov. 27, 1915.

Flotation of Silver-Lead Mineral at a New South Wales Mine. H. Hardy Smith. (6) Dec. 11, 1915.

Flotation at Gold Hunter Mill. (6) Dec. 25, 1915.

New Ideas about Flotation. (6) Jan. 1, 1916.

Progress of Flotation in 1915. H. A. Megraw. (6) Jan. 8, 1916.

Flotation Replaces Cyanide. R. W. Smith. (6) Jan. 15, 1916.

Flotation of the Consolidated Arizona Smelting Co., Humboldt, Arizona. (3) Dec. 1, 1915.

The Cyanide Plant of the Baker Mines Co., Cornucopia, Oregon. R. M. Keeney. (3) Dec. 15, 1915.

Electrolytic Antimony Refining. A. G. Betts. (3) Nov. 15, 1915.

The Electrolytic Precipitation of Gold, Silver, and Copper from Cyanide Solutions. G. H. Clevenger. (3) Serial commenced Nov. 1, 1915.

Mechanical Principles of the Blast Furnace, Part II. J. E. Johnson. (3) Jan. 15, 1916.

Blast Furnace Smelting of Cyanide Precipitate. R. Chauvenet. (3) Jan. 15, 1916.

Treatment of Silver Furnace Fume by the Cottrell Process. C. H. Aldrich. (5) Dec. 11, 1915.

Flotation: its Progress and its Effect upon Mill Design. C. A. Tupper. (5) Jan. 1, 1916.

Cyanide Treatment of Flotation Concentrate. Chas. Butters and J. E. Clennell. (8) Nov. 20, 1915.

Cyanidation in Western Australia. V. F. Stanley Low. (8) Nov. 27, 1915.

Precipitation with Zinc-Lead. J. A. Carpenter. (8) Dec. 11, 1915.

Testing Ores for the Flotation Process. O. C. Ralston and G. L. Allen. (8) Serial commenced Jan. 1, 1916.

The Washoe Reduction Works, Anaconda. L. S. Austin. (8) Serial commenced Feb. 5, 1916.

Notes on Flotation. J. M. Callow. (29) Bull. No. 108.

Segregation in Gold Bullion. J. H. Hance. (29) Bull. No. 110.

The Behaviour of Stibnite in an Oxidizing Roast. H. O. Hofman. (29) Bull. No. 109.

The Determination of Antimony in the Products Obtained by Roasting Stibnite. W. T. Hall and J. Blatchford. (29) Bull. No. 109.

A Development of Practical Substitutes for Platinum and Its Alloys, with Special Reference to Alloys of Tungsten and Molybdenum. F. A. Fahrenwald. (29) Bull. No. 109.

The Effect of Heat in Cyaniding Gold Ores. E. A. Wraight. (25) Bull. No. 135.

Recovery of Mercury from Residues of Amalgamated Cobalt Ores. E. B. Thornhill. (27) Vol. XVIII.

Cottrell Process of Electrical Precipitation. W. A. Schmidt. (27) Vol. XVIII.

MINING.

Notes of Shrinkage Stoping. E. H. Dickenson and H. J. Volker. (6) Nov. 27, 1915.

Underground Mining Methods of Utah Copper Co. (6) Jan. 29, 1916.

Drilling in Narrow Stopes. P. B. M'Donald. (8) Jan. 1, 1916.

Interpretation of Assay Curves for Drill Holes. Edward H. Perry and A. Locke. (29) Bull. No. 110.

Underground Mining Methods of Utah Copper Co. T. S. Carnahan. (29) Bull. No. 109.

Broken Hill Underground Mining Methods. E. J. Horwood. (29) Bull. No. 109.

MISCELLANEOUS.

Sluicing Methods at Fairbanks. H. I. Ellis. (6) Dec. 18, 1915.

Drill Steel and its Treatment. E. M. Weston. (6) Dec. 18, 1915.

Californian Dredge with Two Tailings Stackers. H. H. Eddy. (6) Jan. 22, 1916.

Solution of Smoke, Fume, and Dust Problems by Electrical Precipitation. H. Bradley. (3) Dec. 1, 1915.

Cause, Prevention, and Method of Dealing with Underground Fires. S. A. Jones. (27) Vol. XVIII.

Notes on Mine Ventilation. J. Shanks. (27) Vol. XVIII.

Australasian Institute of Mining Engineers.

EXTRACTS FROM RULES.

III.—MEMBERSHIP.

1. The Institute shall consist of three classes, viz., Members, Associate Members and Students.

2. MEMBERS shall be persons not under 25 years of age who, having occupied during a sufficient period a responsible position in connection with the practice or science of mining or metallurgy, may be considered by the Council to be qualified for election.

(NOTE.—The term "responsible position" covers persons in full charge of important mining or metallurgical operations, in practice as consulting Mining Engineers, engaged in teaching geology, mining or metallurgy in an important institution, or in employment as principal Geologists or Metallurgists in Government Departments and leading Mining Corporations, or Metallurgists after proper training and experience).

3. ASSOCIATE MEMBERS shall be persons not under 21 years of age engaged in work connected with the practice or science of mining, but not in the opinion of the Council in sufficiently important or responsible positions to qualify them for election as members.

4. STUDENTS shall be persons not under 16 nor over 25 years of age, They must be either students of some recognized School of Mining or occupy some subordinate position connected with the practice or science of mining or metallurgy.

IV.—ELECTION OF MEMBERS.

1. Any person desirous of becoming a Member or Associate Member shall be nominated by not less than three Members or Associate Members, provided that one of the nominators certifies to personal knowledge of the candidate's qualifications and suitability and that one at least is a Member.

2. Each person desirous of becoming a Student shall furnish the Council with satisfactory evidence of his connection with mining, and, if approved by the Council, shall thereupon be admitted by the Council; provided that the Council may, after due enquiry, remove any Student from membership.

3. Each nomination of a candidate as Member or Associate Member shall set out distinctly the name, usual residence, and full qualification of the person nominated, shall bear the written consent of the candidate to such nomination, and shall be forwarded to the Secretary, who shall forthwith submit it to the Council. The Council shall determine whether the qualifications of a candidate are sufficient according to the Rules and if approved, may submit the nomination to Members and Associate Members for confidential report. The Council may elect the candidate to the class for which he is eligible provided that, in the event of a candidate's nomination having been submitted to Members and Associate Members for confidential report, the Council shall not proceed with his election for six weeks after such submission.

7. Associate Members desirous of being transferred to the class of Members must make application for transfer to the Council. Such application must be signed by at least three Members and must give fully the evidence upon which the applicant relies.

8. Students may not continue to be connected with the Institute after attaining the age of 25 years. They may make application for admission as Associate Members on attaining the age of 21 years, or as Members on attaining the age of 25 years.

VI.—SUBSCRIPTION.

1. On election each Member shall pay an Entrance Fee of Two Guineas and each Associate Member One Guinea and a Half; on transfer from Associate Member to Member, each Associate Member shall pay a fee of Half-a-Guinea.

2. Each Member shall pay an Annual Subscription of Two Guineas, each Associate Member, One Guinea and a Half; and each Student, Half-a-Guinea.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 22.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

30th JUNE, 1916.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

LIST OF OFFICERS, 1916.

President:

Robert C. Sticht, Queenstown, Tas.

Vice-Presidents:

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D. B. Waters, Dunedin, N.Z.

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Lindesay C. Clark	Tasmania
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William Poole	New South Wales
H. H. Schlapp	Victoria
R. W. Chapman	South Australia
E. G. Banks	New Zealand
W. H. Corbould	Queensland
C. F. Courtney	New South Wales
H. Lipson Hancock	South Australia
H. Herman	Victoria
E. W. Skeats	Victoria
John W. Moule	Queensland
G. D. Delprat	New South Wales
R. S. Black	Western Australia
Richard Hamilton	Western Australia
A. S. Kenyon	Victoria
Geo H. Broome	Victoria
George Weir	New South Wales
George C. Klug	Western Australia

Secretary:

D. L. Stirling, Melbourne, Vic.

LOCAL CORRESPONDENTS:

South Australia	James P. Wood, Adelaide.
New South Wales	F. Danvers Power, Sydney.
New Zealand	D. B. Waters, Dunedin.
Tasmania	Russell M. Murray, Mount Lyell.
North Queensland	Murray Russell, Cloncurry.

BRANCH SECRETARIES:

Kalgoorlie	F. W. R. Godden, Ivanhoe Gold Corporation, Boulder, W.A.
Broken Hill	J. M. Bridge, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Mount Morgan	B. G. Patterson, Mount Morgan, Q.

The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 Swanston Street, Melbourne, Victoria.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Summary).

APRIL 3RD, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £25 were passed for payment.

Nomination of Mr. C. M. Lyons as a Member was approved.

It was reported that the Institute, in conjunction with kindred Associations, had agreed to conjointly oppose the registration of the Union of Architects, Engineers and Surveyors on account of the title assumed.

Other routine business was transacted.

MAY 8TH, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £30 were passed for payment.

Nomination of Mr. Godfrey Stevenson as a Member was submitted and deferred.

A paper by Mr. W. Shellshear entitled "Notes on the Flotation of Gold and Copper Ores, Mt. Morgan, Queensland" was referred to the Publication Committee.

It was reported that the application for the registration of the proposed Union of Architects, Engineers and Surveyors had been refused.

Further consideration was given to the question of Institute representation on Boards and Faculties.

JUNE 1ST, 1916—1 P.M

The Secretary's report was presented and accounts to the amount of £119, including printing Proceedings £54, Broken Hill Branch £25 and Rent £12, were passed for payment.

Mr. G. B. L. Symonds was admitted as a Student.

It was decided that £250 worth of Commonwealth Inscribed Stock be purchased.

Mr. H. H. Schlapp was appointed to represent the Institute on the Faculty of Engineering at the Melbourne University and progress was reported regarding Institute representation on other Boards and Faculties.

It was resolved that the Institute be represented at the forthcoming Conference on Australian Timbers.

It was decided that members of Council be supplied with bound copies of Proceedings, commencing with No. 5 (New Series).

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

PUBLICATIONS.

Papers, etc. appearing in this number are :—

PAPERS—Notes on the Flotation of Gold and Copper Ores, Mt. Morgan, Queensland. By W. Shellshear.

Analysis of Mineral Waters by Volumetric Methods. By E. Broughton Jensen.

DISCUSSION—The Organization and Equipment of a Mine-Rescue Station (J. C. Coldham). By Hartwell Conder.

ON ACTIVE SERVICE.

The following have notified their enlistment since the issue of the March Proceedings :—

BARKER, C. S.
BURBIDGE, PERCY
HAMMEL, H. E.
HUGHES, ERIC W.
McCALLUM, N. S. KELLIE
PEAT, JAMES
RAE, CECIL
WENTWORTH, D'ARCY
WILLIAMS, O. B.

Notes concerning Members on Active Service.

W. T. Anderson, of Sydney, is a 2nd Lieutenant in the Australian Mining Corps.

Dr. James Mackintosh Bell (Past President) is a Major in the Canadian Highlanders.

R. G. Casey, Jun., who was a Staff Officer in Egypt and Gallipoli, was invalided to Gibraltar and England. He subsequently returned to Egypt and is now in France as a General Staff Officer of the Third Grade (Captain) in the Australian Forces.

Leslie J. Coulter, of Mount Lyell, is a Major and second in command of the Australian Mining Corps.

Stanley B. Hunter, of the Mines Department, Victoria, is a Captain on the technical staff of the Australian Mining Corps.

Keith B. Lewis, as 2nd Lieutenant in the 22nd Battalion Infantry, took part in the Gallipoli campaign. He was invalided

to Malta, thence to Egypt and Australia. He has returned to the front as a Lieutenant in the 5th Tunnelling Company, Australian Mining Corps.

James McBryde, of the Cock's Pioneer Mine, Victoria, is now a Lieutenant in the 1st Field Company of Engineers, 1st Australian Division, abroad.

Kenneth A. Mickle, of Melbourne, and late of the Burma Mines Limited, enlisted in England in the Royal Garrison Artillery, is now in France and holds a commission as 2nd Lieutenant.

Cecil Rae is a Lieutenant in the Indian Army.

W. A. Turner is supervising construction of roads, railways, pipe lines and water installations at Kantara, Egypt, under Captain Colson of the Royal Engineers.

Professor D. B. Waters (Vice President) is a Captain in the New Zealand Engineer Tunnelling Company, now at the front.

The Bandmaster of the Australian Mining Corps (Mr. W. S. Curtis) reported in April that the Band, with instruments presented through the Institute, had been voted a source of pleasure to the 2500 troops on board the transport. He also reports a perfect voyage and an entire absence of sickness amongst the men.

OBITUARY.

Henry Rosales, at his residence, Grand View Grove, Armadale, on June 3, at the age of 96 years. Born in Spain and educated as a mining engineer in France and Germany, he came to Victoria in the late fifties where he followed his profession first at Ballarat and Clunes. He then became manager of the Walhalla Gold Mining Co. Mr. Rosales was a Fellow of the Australasian Institute of Mining Engineers and took a keen interest in its affairs. He leaves a widow and many sincere friends to regret his death.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY
From 31st March, 1916, to 30th June, 1916.

Australian Mining Standard	weekly	Melbourne
Australian Mining and Engineering Review	monthly	Melbourne
Engineering and Mining Journal	weekly	New York
Iron and Coal Trades Review	weekly	London
Mining Journal	weekly	London
Mining and Scientific Press	weekly	San Francisco
The Colliery Engineer	monthly	Scranton, Pa.
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry: Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa: Journal	monthly	Johannesburg
Franklin Institute: Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers: Journal	monthly	London
Metallurgical and Chemical Engineering	monthly	New York
Chamber of Mines of Victoria: Monthly Mining Report		Melbourne
Chamber of Mines of Western Australia: Journal	monthly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines: Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines: Monthly Analysis of Production		Johannesburg
Rhodesia Chamber of Mines: Report of Executive	monthly	Bulawayo
Royal Society of Victoria: Proceedings, Vol. XXVIII., Part 2		Melbourne
Department of Mines, Queensland: Geological Survey, Publication, No. 253		Brisbane

Department of Mines, Western Australia :	
Geological Survey, Bulletin, No. 63 - -	Perth
Department of Mines, Tasmania :	
Geological Survey, Bulletin, Nos. 22-24	
Mineral Resources, No. 1, Parts 1 and 2 -	Hobart
Institution of Mining and Metallurgy :	
Bulletins, Nos. 137-139 ... - -	London
Geological Society :	
Quarterly Journal, Vol. LXXI., No. 282 -	London
Institution of Mining Engineers :	
Transactions, Vol. L., Part 4 ; Vol. LI., Part 1	London
Annales des Mines :	
Vol. III. and IV., 1915 - - -	Paris
Société Des Ingenieurs Civils De France :	
Memoirs, Series 7, Nos. 10-12	
Procès Verbal De La Séance Du 25 Fevrier 1916,	
No. 1 and 2 - - - -	Paris
Geological Institution of the University of Upsala :	
Bulletin, Vol. XIII, 1915 - - -	Upsala
Geological Survey of India :	
Record, Vol. XLV., Part 4 - - -	Calcutta
Mysore Geological Department : Records, Vol. XIV. -	Bangalore
Department of Mines, Canada :	
Memoirs, Nos. 34, 50, 58, 60, 63, 72, 76, 81	
Museum Bulletin, No. 22 - - -	Ottawa
American Institute of Mining Engineers :	
Bulletins, Nos. 111-113 - - -	New York
Geological Survey of Alabama : Bulletin, No. 16, 1916	Alabama

RECENT ARTICLES ON MINING MATTERS.

(31st March, 1916, to 30th June, 1916).

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly; mth., monthly.

(1) *The Australian Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk. 6d.

(5) *Mining and Engineering World*, Chicago, wk., 10c.

(6) *The Engineering and Mining Journal*, New York, wk., 15c.

(7) *The Colliery Engineer*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining and Scientific Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

(11) *Publications*, Department of Mines, Sydney, New South Wales.

(12) *Publications*, Department of Mines, Adelaide, South Australia.

(13) *Publications*, Department of Mines, Brisbane, Queensland.

(14) *Publications*, Department of Mines, Perth, Western Australia.

(15) *Publications*, Department of Mines, Hobart, Tasmania.

(16) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(17) *Publications*, Bureau of Mines, Toronto, Ontario.

(18) *Publications*, Geological Survey of India, Calcutta.

(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(24) *Quarterly Journal*, Geological Society, London.

(25) *Transactions*, Inst. Mining and Metallurgy, London, E.C.

(26) *Transactions*, Inst. Min. Eng., London.

(27) *Journal*, Canadian Mining Inst., Ottawa, Ontario.

(28) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(29) *Transactions*, Am. Inst. of Min. Eng., New York City.

(30) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(31) *Journal*, Franklin Inst., Philadelphia, Pa.

(32) *Australian Mining and Engineering Review*, Melbourne, Vic., mth., 6d.

(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins*, Société des Ingénieurs Civils, Paris.

(35) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(36) *Publications*, Iron and Steel Institute, London.

(37) *Proceedings*, Inst. of Mech. Eng., London.

(38) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

(41) *Publications*, Department of Mines, Wellington, N.Z.

(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(44) *Proceedings*, Geologists' Association, London.

LIST OF ARTICLES

GEOLOGICAL.

The Conglomerates of the Witwatersrand. E. T. Mellor. (28) Feb., 1916.
Gold–Quartz Replacements in Intrusive Rock. J. F. M'Lennan. (5) Feb. 19, 1916.
Geology of Tungsten Deposits. J. J. Runner. (8) March 18, 1916.
Geology and Mineral Resources of the Yilgarn Goldfield. T. Blatchford. (14) Bull. No. 63.

MECHANICAL.

Chrome Ore as Lining for Reverberatory Furnaces. Edgar Hall. (6) Feb. 5, 1916.
Coal-Dust Firing in Reverberatory Furnaces. C. R. Kuzell. (6) Feb. 12, 1916.
High-Speed Air Compressors for Mining Work. J. M. Walshe. (26) Vol. LI., Part 1.

METALLURGICAL.

Electrolytic Zinc. W. R. Ingalls. (6) March 4, 1916.
Sedimentation and Flocculation. E. E. Free. (6) March 4, 1916, also March 18, 1916.
Molecular Forces in Flotation. D. E. Norris. (8) Feb. 12, 1916.
Smelting Flotation Concentrate. (8) Feb. 12, 1916.
Refining Cupriferous Precipitate. J. A. Pearce. (8) Feb. 19, 1916.
Flotation Principles. C. T. Durell. (8) Feb. 19, 1916.
On the Science of a Froth. W. H. Coghill. (8) Feb. 26, 1916.
The Flotation Process. T. A. Rickard. (8) Serial commenced March 4, 1916.
Electro-Statics of Flotation. F. A. Fahrenwald. (8) March 11, 1916.
Electrolytic Zinc. H. A. B. Motherwell. (8) March 18, 1916.
The Hydrometallurgical Treatment of Complex Gold and Silver Ores. G. H. Clevenger. (3) Feb. 15, 1916.
Universal Flotation Theory. C. Terry Durell. (3) March 1, 1916.
Flotation Concentration at Anaconda. F. Laist and A. E. Wiggin. (29) Bull., March, 1916.
Laboratory Methods for Determining the Capacities of Slime-Settling Tanks. H. S. Cole and G. H. Clevenger. (29) Bull., March, 1916.
Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces. H. P. Howland. (29) Bull., March, 1916.
Flotation Symposium at Ottawa. (3) March 15, 1916.
Cyaniding by Continuous Decantation at two Nevada Silver Mills. (3) April 15, 1916.
The Electrolytic Determination of Copper in Copper-Manganese. E. D. Koepping. (3) April 15, 1916.
Burdening the Blast Furnace. J. E. Johnson. (3) April 15, 1916.
Rate of Slimes Settling. E. E. Free. (6) April 15, 1916.
Milling of Tungsten Ores. J. F. Magee. (6) April 22, 1916.
Seasoning Reverberatory Furnaces. E. C. King. (6) April 22, 1916.
The Control of Slimes. O. C. Ralston. (6) April 29, 1916.
Concentration of Zinc Ore in Wisconsin. H. P. Wherry. (8) April 22, 1916.
Oils for Flotation. C. Y. Clayton and C. E. Peterson. (8) April 22, 1916.
Working Data in Electrolytic Precipitation. P. H. Crawford. (8) April 29, 1916.
Double Roasting Process at East Helena. (8) May 6, 1916.
Flotation Tests on Joplin Lead and Zinc Ores. (5) April 15, 1916.
Antimony Smelting in the Hunan Province, South China. A. S. Wheeler. (5) April 15, 1916.
A Simple and Rapid Assay of Lead. (5) April 22, 1916.
Braden Roasting and Sulphuric Acid Plant. J. B. Wise. (5) April 29, 1916.
Some Notes on the Effect of Lead Salts and of Varying Degree of Alkalinity on the Solvent Power of Cyanide Solution of Gold. H. R. Edmands. (42) April, 1916.
Interfacial Tension in Flotation. H. J. Stander. (6) March 25, 1916.
High Sulphur Does Not Injure Open-Hearth Steel. (6) April 1, 1916.
Precipitating Action of Carbonaceous Shale in Cyanide Solution. P. W. Avery. (8) April 8, 1916.
Machinery for Cyaniding Flotation Concentrate. A. E. Drucker. (8) April 8, 1916.
A Simple and Rapid Assay of Lead. G. Torossian. (43) April, 1916.
The Operations of the Blast Furnace. J. E. Johnson. (3) April 1, 1916.
The Available Hearth Heat of the Blast Furnace. A. L. Feild. (3) April 1, 1916.

Some Sources of Error in the Iodometric Determination of Copper. Carl E. Smith. (3) April 1, 1916.

The Metallurgical Disposal of Flotation Concentrates. R. J. Anderson. (3) April 1, 1916.

Mass Screening with Flat Screens. E. S. Wiard. (3) April 1, 1916.

A Proposed Quick Analytical Method for Determining the Zinc in Retort Residues or Electric Zinc Furnace Slags. W. M'A. Johnson. (3) April 1, 1916.

Decomposition and Reduction of Lead Sulphate at Elevated Temperatures. W. Mostowitsch. (29) May, 1916.

A Combined Hydraulic and Mechanical Classifier. M. G. F. Sohnlein. (29) April, 1916.

MINING.

Stoping Methods. F. W. Sperr. (8) Feb. 19, 1916.

Mitchell Top-Slice and Caving System. R. H. Dickson. (6) March 25, 1916.

MISCELLANEOUS.

Recent Progress in Electrical Smoke Precipitation. F. G. Cottrell. (6) Feb. 26, 1916.

Tungsten and Molybdenite. (15) Mineral Resources. No. 1.

Zinc-Lead Sulphide Deposits of the Rear-Rosebery District. Loftus Hill. (15) Bull. No. 22.

Rifling of Diamond-Drill Cores. W. R. Crane. (29) May, 1916.

SUBSCRIPTIONS.

Members are reminded that Subscriptions for **1916** are due, and are requested to forward same as early as possible.

GUIDE TO CONTRIBUTORS OF PAPERS.

It is requested that writing be confined to *one* side of the paper ; that it be legible with particular care regarding foreign words; and that abbreviations, references, &c., be made in accordance with the subjoined rules and examples:—

Pounds, shillings, and pence, £ s. d. ; dollars and cents, \$, c.
Grains, gr., pennyweights, dwt. ; ounces, oz. ; drachms, dr. ; pounds, lb.
Quarters, qr ; hundredweights, cwt.; gallons, gal.
Grammes, grm. ; kilogrammes, kg ; millimetres, mm.; milligrammes, mg.
Centimetres, cm. ; cubic centimetres, cc.; metres, m.
Inches, in.; feet, ft.: yards, yd.: fathoms, fath.
Square inches, etc , sq. in.; cubic inches, etc., cub. in.
Diameter, diam.; revolutions, rev.; revolutions per minute, r.p.m.
Horse power, h.p.; indicated horse power, i.h.p.; brake horse power, b.h.p.
Candle power, c p.
British thermal units, B.Th.U.
High pressure, H.P. ; low pressure, L.P.
Ampere, amp.: kilowatts, kw.
Percentages, %; degrees, $^{\circ}$; specific gravity, sp. gr.
Company Limited, Co. Ltd.; and Company, & Co.
Temperatures to be given in Fahrenheit, thus : 10° Fahr.
Figures not exceeding four, unless in column with others exceeding four, to be without comma, thus : 1907.

References to be placed in foot-notes, giving title in italics, thus : *Alfred James. *Cyanide Practice*, p. 94 ; † H. Brown. *Mines and Minerals*, vol. xiii., p. 130; ‡ *Trans Aust. Inst. M.E.*, vol. x, pp. 98-189.

Quotations to be indicated by inverted commas, and when lengthy, to be set in smaller type, with inverted commas at beginning and end only. Foreign terms to be in italic.

Localisms to be in inverted commas with their ordinary technical definition in parenthesis.

Drawings (on tracing cloth if possible), photographs (unmounted glossy-surface bromides) or other glossy-surface prints, &c., both for exhibition and for publication, are most desirable with almost every communication, and should always be on separate paper from the MS. Blue-prints only to be forwarded when original plans or tracings are not procurable. When reference letters are used in drawings for purposes of description, they are best thus : *a b c*, etc. Lettering and figuring on drawings should be large to allow of reduction to page size whenever possible. All lines should be black and firm. Each drawing should be accompanied by scale thus :—

N.B.—When in doubt about any point, it would be advisable to communicate with the Secretary

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 23.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

30th SEPTEMBER, 1916.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

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North Queensland	MURRAY RUSSELL, Cloncurry.

BRANCH SECRETARIES:

Kalgoorlie	F. W. R. GODDEN, Ivanhoe Gold Corporation, Boulder, W.A.
Broken Hill	J. M. BRIDGE, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Mount Morgan	B. G. PATTERSON, Mount Morgan, Q.

The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 SWANSTON STREET, MELBOURNE, VICTORIA.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Summary).

JULY 3RD, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £33 were passed for payment.

It was decided that £350 worth of Commonwealth Inscribed Stock be purchased under the Extended Instalment System, instead of £250 worth as previously resolved.

A report of progress in regard to Institute representation on Board and Faculties controlling Mining Education was submitted and showed that a number of engineering schools and Universities had agreed to the desired representation.

Enquiries were directed to be made in regard to the reported proposal by the Federal Government, viz:—that only qualified mining engineers should be allowed to sign prospectuses, etc., of mining companies. A committee was appointed to safeguard the interests of the Institute in the matter.

Other routine business was transacted.

AUGUST 7TH, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £89, including printing Proceedings £54, were passed for payment.

Applications from Messrs. Denis L. Ryan and Godfrey Stevenson for admission as Associate Members were approved.

The following were elected to the Institute:—As Members—Messrs. Ernest Howard Greig and Charles M. Lyons. Transfer from Associate Member to Member—Messrs. Stanley Robert Mitchell and Francis Guy Wilson. As Associate Member—Mr. George David Reid.

Mr. H. Herman was appointed chairman of the Executive Committee during Mr. Schlapp's absence from Australia.

Other routine business was transacted.

SEPTEMBER 4TH, 1916—1 P.M

The Secretary's report was presented and accounts to the amount of £22 were passed for payment.

Resolved that the Annual Meeting, 1917, for the election of officers and to receive the Annual Report and Balance Sheet be held on Friday, 26th January, 1917, at 1 p.m.

Consideration was also given to the question of the locality of the First Ordinary Meeting for 1917.

A letter from the Advisory Committee of Science and Industry requesting particulars of any technical difficulties and disabilities which affect the development and progress of the mining industry was submitted. It was resolved that the attention of members be directed to the letter through the medium of the "Proceedings."

Resolved that special application be made for payment of subscriptions in arrears.

A paper by Mr. L. H. Ower on "Cyclometer Surveys" was referred to the Publication Committee.

Other routine business was transacted.

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Applications for admission to the institute have been received from the following, whose names are herewith submitted to Members for confidential report:—Members—Messrs. EDWARD JOHN SCOBLE, Kikurangi, Whangarie, N.Z. and T. IMANAGA, Müke, Japan.

Since the publication in proceedings No. 19 of a supplementary list of Members, the following have been admitted:—

MEMBERS.

NAMES AND ADDRESSES.

GREIG, ERNEST HOWARD, Wain Tu, Upper Burma.

LYONS, GHARLES M., Tavoy, Burma.

MITCHELL, STANLEY ROBERT, Canterbury Road, Box Hill, Vic. (transfer from Associate Member).

WILSON, FRANCIS GREY, Emmaville, N.S.W. (transfer from Associate Member).

ASSOCIATE MEMBER.

REID, GEORGE DAVID, Avoca, Vic.

PUBLICATIONS.

The paper published in this issue is entitled:—

Clyclometer Surveys. By Leslie H. Ower.

ON ACTIVE SERVICE.

The following have notified their enlistment since the issue of the June Proceedings:—

BLOOM, L. | DUNSTAN, B. | MACKENZIE, D. C.

Honors Won.

Major Leslie J. Coulter has been awarded the Distinguished Service Order, When a push-pipe failed to explode he went out under heavy fire and blew up the exposed portion of the push-pipe. Later on he tried to light the fuse farther down the sap. He was wounded but refused to be removed until the push-pipe exploded.

ANNUAL MEETING, 1917.

The Annual Meeting, 1917, will be held at the rooms, 57-59 Swanston Street, Melbourne, on Friday, 26th January, 1917, at 1 p.m. The business of the meeting will include the presentation of the Annual Report and Balance Sheet for 1916, and the election of officers to fill the annual vacancies. The officers retiring are :—President, Mr. Robt. C. Sticht; Vice-President, Professor D. B. Waters; Councillors, Messrs. James Hebbard, F. Danvers Power. Lindesay C. Clarke, William Poole, H. H. Schlapp and Professor Chapman. Nominations to fill the vacancies thus occurring should reach the Secretary forty days before the Annual Meeting.

ADVISORY COUNCIL OF SCIENCE AND INDUSTRY.

The Victorian Committee of the Federal Advisory Council of Science and Industry has been requested by the Executive Committee of the Council to ascertain :—

1. Whether there is any industry in Victoria which has encountered technical difficulties and disabilities which affect its development and progress.
2. Whether any particular firm or any person engaged in a given industry has encountered and failed to overcome such difficulties and disabilities.
3. Whether there is any industry which could advantageously be established in Victoria either :—
   (*a*) *Primary*—Such as the production of raw material, or utilization of products not yet turned to account.
   (*b*) *Secondary*—Such as the utilization of by-products from existing undustries.

Members are invited to forward to the Council of the Institute any information in the direction indicated above.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

FROM 30TH JUNE, 1916, TO 30TH SEPTEMBER, 1916.

Australian Mining Standard - -	weekly -	Melbourne
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Transvaal Chamber of Mines : Monthly Analysis of Production	-	Johannesburg
Rhodesia Chamber of Mines : Report of Executive - -	monthly -	Bulawayo

Department of Mines, Victoria :
Geological Survey, Bulletin, Nos. 37 and 38 - Melbourne
Department of Mines, New South Wales :
Mineral Resources, No. 23 - - Sydney
Royal Society of New South Wales :
Journal and Proceedings, Vol. XLIX., Part 3 - Sydney
Royal Society of Queensland :
Proceedings, Vol. XXVII., Parts 1 and 2 - Brisbane
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Review of Mining Operations to Dec. 31, 1915 - Adelaide
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Transactions, Vol. XXIV., Bulletins, Nos. 140-142 London
Institution of Chemistry of Great Britain and Ireland :
Proceedings, 1916, Parts 1 and 2 - - London
Institution of Mining Engineers :
Transactions. Vol. LI., Parts 2-5 - - London
Real Academia de Ciencas :
Revista, Vol. XIII., Nos. 1-12; Vol. XIV., Nos. 1-11 - - - - Madrid
Societa Geologica Italiana :
Bulletin, Vol. XXXV., Part 1 - Rome
Geological Survey of India :
Records, Vol. XLVII., Parts 1 and 2 - Calcutta
Department of Mines, Canada :
Memoirs, Nos. 55, 77, 79
Mineral Production of Canada, 1915
Bulletins, Nos. 12 and 13
Annual Report of Mineral Production, 1914
Summary Report Geological Survey, 1915 - Ottawa
Bureau of Mines, Canada :
24th Report, Parts 1-3, Vol. XXIV., Part 3, 1916 Toronto
American Institute of Mining Engineers :
Bulletins, Nos. 114-116 - - - New York
University of California :
Bulletin, Vol. 8, Nos. 1-22 ; Vol. 9, Nos. 1-18 ; Vol. 10, No. 1 - - - - Berkeley

United States Geological Survey:
Water Supply Papers, Nos. 340 L, 351, 352, 355, 359, 370-372, 375 C, 375 E, 376-379, 385, 388 and 400 A
Bulletins, Nos. 565, 568, 573, 591, 603-608, 615-617, 620 B-620 N, 621 A-621 D, 621 G-621 O
Professional Papers, Nos. 95 E-95 I
Mineral Resources, Various Papers - - Washington
University of Illinois: Bulletins, Vol. XII., Nos. 84-86 Urbana

RECENT ARTICLES ON MINING MATTERS.

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(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(44) *Proceedings*, Geologists' Association, London.

LIST OF ARTICLES

MECHANICAL.

Belt and Bucket Elevators. A. O. Gates. (6) July 1, 1916.
Air Lifts at a Cyanide Plant. P. W. Gaebelein. (6) July 1, 1916.
Safety Cage Gates. R. F. Smith. (6) July 8, 1916.
Diesel Engines *versus* Steam Turbines for Mine Power Plants. H. Haas. (29) Bull. No. 115.
Convenient Inspection Gauge for Hand Drills. C. C. Phelps. (6) May 27, 1916.
Portable Mining Equipment for Prospects. L. A. and W. C. Rehfuss. (6) June 10, 1916.
Drill and Tool Sharpening Shop at Copper Queen Mine. "G. S." (6) June 24, 1916.

METALLURGICAL.

The Use of Belt Conveyors. (6) July 1, 1916.
The Dry Chlorination of Complex Ores. S. A. Ionides. (8) May 27, 1916.
Effect of Black Slate on Cyanidation. H. Fischer. (8) May 20, 1916.
Fine Grinding: Stamps and Ball Mills. H. Hanson. (8) May 13, 1916.
Quicksilver Reduction. H. Lang. (8) May 13, 1916.
Soap as a Frothing Agent in Flotation. M. H. Thronberry. (8) May 13, 1916.
Specific Gravity Method for Tungsten Analysis. J. J. Runner. (8) July 1, 1916.
The Theory of Flotation. H. Hardy Smith. (8) July 1, 1916.
The Flotation of Minerals. R. J. Anderson. (8) July 3, 1916.
Copper Metallurgy at Garfield, Utah. L. O. Howard. (8) July 8, 1916.
Determination of Antimony. H. R. Layng. (8) July 8, 1916.
Discrepancies in Cyanidation. E. Shaw. (8) July 15, 1916.
An Improved Pneumatic Flotation Machine. J. M. Hyde. (8) Aug. 5, 1916.
Simple Tests for Potash. W. B. Hicks. (8) Aug. 5, 1916.
Milling Practice at the Santa Gertrudis. Hugh Rose. (8) Aug. 12, 1916.
Estimating Metallic Aluminium in Aluminium Dust. J. E. Clennell. (6) May 6, 1916.
How Flotation Works. G. D. Van Arsdale. (6) May 13, 1916.
Tin Smelting at Perth Amboy, N.J. R. H. Vail. (6) May 27, 1916.
Control of Ore Slimes. O. C. Ralston. (6) Serial commenced April 29, 1916.
The Roasting of Blende. M. de Lummen. (6) June 10, 1916.
The Sintering of Flotation Concentrates. B. Magnus. (6) June 10, 1916.
Properties of Slime Cakes. E. E. Free. (6) Serial commenced June 17, 1916.
Electrothermic Zinc Smelting. (6) June 17, 1916.
Automatic Car Return. (6) June 24, 1916.
Repairing Wilfley Table Decks. W. L. Zeigler. (6) June 24, 1916.
Cyaniding Copper-Bearing Ores. G. W. Gaebelein. (6) July 1, 1916.
Flotation of Flour Gold. R. W. Smith. (6) July 1, 1916.
Some Notes on Flotation. R. C. Canby. (6) July 1, 1916.
Froths Formed by Flotation. W. A. Mueller. (6) July 1, 1916.
Bunker Hill and Sullivan Milling Data. R. S. Handy. (6) July 1, 1916.
Use of Oils in Flotation. H. H. Megraw (6) July 1, 1916.
Drilling and Analysis of Copper Ores. A. J. Sale. (6) July 8, 1916.
Separating Wolfram from Tin. A. Grossberg. (6) July 15, 1916.
Propeller Slimes Agitator. R. Wheeler. (6) July 15, 1916.
Operating a Small Copper Blast Furnace. A. Bregman. (6) July 22, 1916.
The Engle Furnace for Re-distilling Spelter. R. H. Engle. (6) July 29, 1916.
Flotation and Cyanidation. (3) May 15, 1916.
Electrolysis of Alkaline Solutions of Potassium Sulphocyanate. W. J. Crook and A. Thiel. (3) May 15, 1916.
Blast Furnace Operation. J. E. Johnson. (3) May 15, 1916.
The Roitzheim-Remy Continuous Zinc Distillation Process. M. Liebig. (3) June 1, 1916.
The Distribution of Silver Between Metallic Lead and Litharge-Containing Slags. E. Dudley. (3) June 1, 1916.
The Distribution of the Charge Column and the Ascending Gas Column. J. E. Johnson. (3) Serial commenced June 1, 1916.
An Electric Arc Furnace for Laboratory Use. O. P. Watts. (3) June 15, 1916.
Coke as a Refining Agent in the Electric Smelting Furnace. R. C. Gosrow. (3) June 15, 1916.

Extraction of Gold and Silver from Matte by Lead. W. Mostowitsch. (3) June 15, 1916.
Sources of Metal Loss in Copper Refining. L. Addicks. (3) July 1, 1916.
The Rate of Driving the Blast Furnace. J. E. Johnson. (3) July 1, 1916.
Flotation Experiments on a Joplin Tailing. W. A. Whitaker, G. Belchic, R. Neal, and H. L. Van Velzer. (3) Aug. 1, 1916.
Flotation of Oxidized Ores. O. C. Ralston and G. L. Allen. (3) Aug. 1, 1916.
The Flotation of Minerals. R. J. Anderson. (29) Bull. No. 115.
The King Process of Refining Copper. (5) June 24, 1916.
Ore-Sampling Conditions in the West. T. R. Woodbridge. (5) July 29, 1916.
Ore-Valuation: How Arrived at. A. M. Plumb. (5) June 8, 1916.
Mechanical Feeding as Applied to Silver-Lead Blast Furnaces. D. Anderson. (5) May 20, 1916.
Determination of Dust Losses at the Copper Queen Reduction Works. J. M. Samuel. (29) Bull. No. 114.

MINING.

Stoping by Branched Raises. F. W. Sperr. (8) May 20, 1916.
Modern Blasting. P. B. M'Donald. (8) May 27, 1916.
Stoping Hard Ground at Miami, Arizona. (8) June 24, 1916.
Blasting Practice at Chuquicamata, Chile. H. W. Moore. (8) July 8, 1916.
Nomenclature of Mining Methods. G. J. Young. (6) July 22, 1916.
Mining Ore from Pillars. H. H. Hodgkinson. (6) July 29, 1916.
Stoping in the Calumet and Arizona Mines. P. D. Wilson. (29) Bull. No. 115.
Stoping Methods of Miami Copper Co. D. H. Scott and E. M. Miami. (29) Bull. No. 114.

MISCELLANEOUS.

A New Dry-House. R. E Tremoureux. (8) June 17, 1916.
Increase in Cost of Mine Supplies. (8) June 17, 1916.
Gold Saving in Dredges. H. D. Smith. (8) Aug. 5, 1916.
Magnesite Production and Markets. S. H. Dolbear. (8) Aug. 12, 1916.
Samples and Their Interpretation. E. H. Dickenson and H. J. Volker. (6) May 27, 1916.
Churn-Drill Prospecting at Morenci, Arizona. W. R. Grunow. (6) June 3, 1916.
Dust Allaying in Rand Mines. A. Cooper Key. (6) June 17, 1916.
Dip and Strike Calculations from Drill Holes. G. E. Burton. (6) July 15, 1916.
Ingenious Special Devices for Tunnel Surveys. (6) July 22, 1916.
Sampling Placer-Gravel Deposits. I. Herr. (6) Aug. 5, 1916.
Cost Accounting in the Construction and Operation of a Copper Smelter. E. E. Thum. (3) June 1, 1916.
Coke Industry of New South Wales. L. F. Harper. (11) Mineral Resources, No. 23.

x1 a

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining Engineers.

NEW SERIES.

No. 24.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

31st DECEMBER, 1916.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF MINING ENGINEERS, MELBOURNE.

Australasian Institute of Mining Engineers.

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Kalgoorlie	F. W. R. Godden, Ivanhoe Gold Corporation, Boulder, W.A.
Broken Hill	J. M. Bridge, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Mount Morgan	B. G. Patterson, Mount Morgan, Q.

The Executive Committee consists of all Members of the Council residing, or for the time being, in Melbourne.

Head Office:

57-59 Swanston Street, Melbourne, Victoria.

Institute Matters.

CONTENTS.

MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Summary).

OCTOBER 2ND, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £34 were passed for payment.

Applications from Messrs. Edward J. Scoble and I. Imanaga for admission as Members were approved.

A report in connection with the proposed conference on Australian Timbers was submitted.

Mr. A. A. Boyd, General Manager, Mount Morgan G.M. Co., was appointed to fill a vacancy on the Council.

The Report of the Committee appointed to consider the question of signatures to prospectuses was submitted and adopted.

NOVEMBER, 6TH, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £52 were passed for payment.

The application of Mr. T. H. Turner for admission as an Associate Member was approved.

Messrs. G. C. Klug, G. H. Broome and H. H. Schlapp were appointed to represent the Institute on the proposed Advisory Board for the School of Mines Department of the Working Men's College, Melbourne.

Further consideration was given to the matter of signatures to prospectuses.

It was reported that the Minister of Home Affairs had under consideration the suggestion that the Institute be officially represented at the opening of the East-West Railway.

Resolved that Members who had received final notices of subscriptions due be dealt with at the December meeting of the Committee.

DECEMBER 18TH, 1916—1 P.M.

The Secretary's report was presented and accounts to the amount of £81, including printing £24, and Rent £12 were passed for payment.

Resolved that the names of Members whose subscriptions were owing for 1914 and to whom final notices had been issued be removed from the register.

Resignations of Mr. Wm. G. Clarke (Member) and Messrs. T. H. Cooke and J. W. Bainbridge (Associate Members) were accepted.

Recommendations in the matter of signatures to prospectuses were submitted and after revision were directed to be forwarded to Members of Council for advice thereon.

Resolved that, if possible, a series of lectures be arranged during 1917 on "Metals and the War."

NOTICES.

The rooms of the Institute are open from 9.30 A.M. to 10 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Application for admission to the Institute as an Associate Member has been received from MR. TOM HEPPINGSTONE TURNER, 23 Ormond Esplanade, Elwood, whose name is herewith submitted to Members for confidential report.

ON ACTIVE SERVICE.

The following have notified their enlistment since the issue of the September Proceedings :—

BULLEN, G. L.
CLAYTON, C. H. J.
DOW, J.
FINLAYSON, R. J. STANLEY
FOXALL, JOHN S.
HARGRAVES, E. L.
HORSLEY, R. D.
LAKELAND, WM. J.
MOORE, K. BYRON
PETHEBRIDGE, H. V.
SMITH, W. F.
WEIGALL, H. S.
WILKINS, LAURENCE G.
WOODWARD, O. H.

Honours Won.

Signaller Frank H. Nicholas was awarded the Military Medal for conspicuous services at the battle of Pozieres in July last.

Lieut. O. H. Woodward has received the Military Cross and promotion for distinguished service at the front.

Killed in Action.

Lieut. James M'Bryde, of the 1st Field Company of Engineers, A.I.F. was killed in the battle of the Somme on 13th September. He was a Graduate of the Sydney University. Prior to enlisting in the early stages of the war he was on the staff of the Cocks Pioneer Mine at Eldorado, Victoria. He served with distinction at Gallipoli, Egypt and France, taking part in the battle of Pozieres.

Notes concerning Men on Active Service.

Captain D. B. Waters (Vice-President) has furnished interesting particulars concerning the life and work of his company (The New Zealand Tunnelling Company) in the North of France.

Members are requested to notify the Secretary of enlistments and to furnish items of interest concerning Members on active service.

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Proceedings, Vol. XCIII., 1916 - - London
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METALLURGICAL.

Molecular Forces and Flotation. W. H. Coghill. (8) Sept. 2, 1916.
Replacing Mortar Blocks. C. Labbe. (6) Aug. 26, 1916.
The Principles of Filtration. D. R. Sperry. (3) Aug. 15, 1916.
The Function of Oil and Acid in Flotation. H. J. Stander. (5) Aug. 19, 1916.
Possibilities in the Wet Treatment of Copper Concentrates. L. Addicks. (29) Bull. No. 117.
Leaching Tests at New Cornelia. H. W. Morse and H. A. Tobelmann. (29) Bull. No. 117.
History of the Flotation Process at Inspiration. R. Gahl. (29) Bull. No. 117.
A New Flotation Oil. M. Adams. (29) Bull. No. 117.
A New Source of Flotative Agents. G. H. Clevenger. (29) Bull. No. 117.
Zinc Smelting in Vertical Retorts. (8) Sept. 9, 1916.
Concentration and Smelting of Vanadium Ore. R. L. Grider. (8) Sept 9, 1916.
Laboratory Investigations Concerning the Reduction of Barium Sulphate to Barium Sulphide. A. E. Wells. (43) Sept., 1916.
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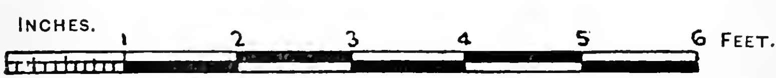

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CONTENTS.

PAPERS—

NOTES ON THE CHEMICAL ASSAY OF TIN ORES.

BY A. M. MATHESON.

THE object of the following notes is to show the difference between fire and chemical assays carried out on a highly pyritic tin ore, and to show the impossibility of estimating mill losses by the vanning and fire method.

The tin occurs in the ore under consideration as cassiterite (SnO_2), the sulphur being mostly combined with Fe, As, Sb, Cu, Pb, etc.

Stannite may occur very sparingly, but, being soluble in the preliminary acid treatment, it would not play a part in either the fire or chemical assay. The preliminary treatment this ore receives is the usual crushing, and concentration with jigs, Wilfley tables, and vanners. The whole of the pyritic concentrate is calcined, and a secondary treatment of the same kind follows. The resulting concentrate for sale averages about—Sn, 71.00 %; Pb, .29 %; S, .31 %; and As, .10 %.

The writer carried out numerous experiments to determine the mill losses, both by fire and chemical assays, and a few are here

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quoted. The fire assay used is that of vanning after digestion, and fusion with cyanide method. The chemical assay is the Pearce–Low method, as follows:—.5 grm. finely-ground ore is digested with aqua-regia and filtered, the filter paper and contents ignited, and fused in a nickel crucible with NaOH.

The cake is dissolved with water and about 30 cc. HCl, transferred to a conical flask, and 50 cc. excess HCl added. A strip of nickel is inserted, and the solution boiled for 1½ hours; the stannic chloride is then reduced to stannous chloride. The nickel is then removed, and a piece of carbonate added to prevent oxidation, the assay cooled and titrated with standard iodine solution with starch indicator.

The three products under consideration in the following assays are:—(1) battery pulp, (2) tails from primary treatment, (3) tails from secondary treatment.

(1) *Battery Pulp.*—

Fire Assay.		Chemical Assay.		Ratio of Chemical to Fire.
%		%		
1.29	..	2.48	..	1.92
1.20	..	2.10	..	1.75
1.70	..	3.14	..	1.84

The above assays are taken from a number done on the same product, but from different samples, and it will be seen that the fire-assay result is a little over 50 % of the chemical-assay value. In the fire assay 100 grm. ore were taken.

(2) *Primary Tails.*—

Fire Assay.		Chemical Assay.		Ratio of Chemical to Fire.
%		%		
.10	..	.84	..	8.40
.19	..	1.03	..	5.42
.12	..	.96	..	8.00

The above assays are taken from a number done on the same product, but from different samples, and it will be seen that the

recovery by fire assay (about 14.5 %) is very much lower than would be expected. However, in making a comparison between this recovery and the recovery of 50 % in the pulp product, it must be remembered that all the easily recoverable oxide has already been taken out by the plant, and only the very fine slime concentrate left for vanning. In the fire assay 100 grm. ore were taken for vanning. Although quite an appreciable amount of oxide was obtained, the resulting tin button was very small indeed (in some cases only about 20 % of the oxide obtained), showing that a further loss took place in the fusion, the tin oxide probably going into the slag in combination with the silica as tannous silicate.

In vanning this tail product it is not possible to get rid of all the silica without making a very great loss of oxide. It becomes questionable, however, whether the loss made in thoroughly cleaning the oxide would be greater than the loss in the fire when smelting in the presence of silica. The writer came to the conclusion, after numerous experiments, that a 100 grm. charge of this product was much too large to handle without a great loss, so a 10 grm. charge was tried, with the following results :—

	%
Chemical assay	.84 Sn.
Oxide obtained	.60 ,,
Metal estimated at 70 %	.42

This shows a recovery of 50 % of the chemical value by vanning. The oxide obtained in this case was considered too small a quantity to smelt, and was estimated at 70 % metal.

Considering the loss that must have taken place in vanning this oxide clean, and the subsequent loss that would have taken place in the fire if smelted, it seems reasonable to suppose that the oxide actually weighed should equal 70 % metal or more. It seemed almost impossible to reach anything like finality in vanning this tail product, a fair proportion of the tin oxide always remaining in suspension even after long settlement. A grading analysis carried out on this product resulted as follows :—

Bulk Chemical Assay, 1.24 %.

Sieve.	Grade.	Assay.	Product.	Tin Percentage.	
+ 20	—	—	—	—	—
+ 40	—	—	—	—	—
+ 60	29.8	1.00	29.8	29.0	—
+ 80	22.5	.80	18.00	17.60	—
+ 100	7.0	.60	4.20	4.10	—
+ 120	17.30	.60	10.40	10.20	—
— 120	23.40	1.72	40.30	39.10	—
Total ..	100.00	1.24	102.70	100.00	1.03

The result of this grading analysis does not check with the bulk assay as well as might be expected, but the low result is most probably due to the mechanical (filtration) loss that takes place, more especially in the finer grades.

(3) *Secondary Tails.*—

Fire Assay.		Chemical Assay.		Ratio of Chemical to Fire.
%		%		
.24	..	.68	..	2.84
.15	..	.48	..	3.20
.18	..	.52	..	2.90

The above assays were taken from a number done on the same product but from different samples, and also show a very low recovery by fire assay (about 33 % of the value as shown by the chemical assay). A 100 grm. charge was used in vanning for the fire assay. This was also a very difficult sample to van, and the same remarks apply as to vanning and fusion losses as in the case of the primary tail sample. When a 10 grm. charge was used in vanning the results were as follows:—

Fire Assay.		Chemical Assay.		Ratio of Chemical to Fire.
%		%		
.40	..	.80	..	2.00
.36	..	.60	..	1.66
.30	..	.56	..	1.87

These assays show a much better recovery by fire, but the vanning and fusion losses are still considerable.

In order to find out what extraction might reasonably be expected from a slime table operating on these two tail products, the following vanning experiments were carried out, the conditions being as near as possible to those obtaining in actual work:—

(1) *Vanning Primary Tail for 15 Minutes Without Acid Treatment, and in Dirty Water.*

(Chemical assay value, 1.24 % Sn.) The pyritic concentrate obtained was acid treated, resulting in oxide equal to .70 %, estimated metal .49 %, showing that the table might be expected to save about 40 % of the tin contents.

(2) *Vanning Secondary Tail for 15 Minutes Without Acid Treatment, and in Dirty Water.*

(Chemical assay value, .88 %.) Oxide obtained .60 %, estimated metal .42 %, showing that the table might be expected to save about 48 % of the tin contents. It might be mentioned here that a Cornish round table is now operating on this tail product, and has made an average saving of 50 %. In these products under consideration it will be noticed that the tin values vary very much indeed; but such is the case, and this condition seems to be consequent on physical changes being encountered in the ore in the mine. The more pyritic the ore becomes the greater are the losses in the tails from primary treatment. The chemical assay records these physical changes very faithfully, while the vanning and fire assay—even if carried out very carefully—fails almost completely to show these changes. Before the introduction of the chemical assay by the writer, these mill losses were estimated by fire assays, and it would appear that they were carried out with more regard to speed than to accuracy. Records taken over some years show little or no variation. The extraction by the plant estimated by these fire assays would be anything from 90 to 97 %, which is altogether too high to expect from even the most up-to-date plant on this class of ore. The

extraction estimated by the chemical assays is very much lower, and certainly much nearer the truth, but does not necessarily show that the plant is not doing good work.

When nickel became unprocurable owing to the war, iron in the form of horseshoe nails had to be used as a reducer in the chemical assay. The results occasionally were low and unreliable, probably due to oxidation taking place. HCl acts much more readily on iron than on nickel, consequently less iron had to be used. Apparently all the acid of the assay was used up in the formation of ferrous chloride, none being left to form CO_2 with the carbonate, and oxidation appears to have taken place.

The following assays were carried out, varying the number of nails used to test this suspected oxidation. These assays were carried out on a pulp sample, the correct assay of which was 3.08 % Sn :—

1.—4 nails	.. 1.48 % Sn.	3.—2 nails	.. 3.08 % Sn.
2.—4 ,,	.. 2.52 % ,,	4.—2 ,,	.. 3.08 % ,,

Oxidation did not always take place with four nails, but the results were low and unreliable. A considerable number of tests were carried out on different samples of a similar product, and a few are here quoted :—Nos. 1, 2, 3, and 4, 3.28 %, 3.24 %, 3.00 %. 3.00 %, using two nails each.

Using two nails only the results became constant and reliable, there being sufficient free acid left in the assay to form CO_2 with the carbonate and prevent oxidation. A further test was made on sale concentrates assaying 70.00 % Sn by the fire assay; (1) one nail gave 63.07 %, and (2) two nails gave 69.74 %.

The following assays on the pulp and tailings show the results of using too much iron :—

1.—Mill pulp—4 nails, .74 %; 2 nails, 1.26 %.

2.—Primary tails—4 nails, .52 %; 2 nails, .70 %.

3.—Secondary tails—4 nails, .36 %; 2 nails, .78 %.

According to some authorities, the chemical assay gives high results, and certain impurities tend to make results high: however, that has not been the experience of the writer. Sale con-

centrates, which must necessarily contain all the heavy impurities of the original ore, concentrated, invariably give slightly low results as compared with the fire assay. These impurities must have, therefore, little effect on the assay of the pulp and tail products.

The Effect of Wolfram.

In particular it has been stated that wolfram interferes with the chemical assay and gives high results. The following test was made on an ore assaying 30.0 % Wo_3 and 4 % Sn. This was treated as an ordinary tin-ore assay, using iron as a reducer, with the result that the assay assumed a deep blue colour shortly after reduction commenced, owing to the formation of a tungsten tungstate. Titration with iodine was impossible. However, the obvious thing to do with an ore carrying any tungsten would be to extract the H_2Wo_4 with ammonia, evaporate the ammonium-tungstate, and estimate as Wo_3, then proceed with the residue on the filter as a tin assay.

The writer does not contend that the chemical assay should take the place of the vanning assay in estimating mine samples. The latter is shorter, and certainly serves as a guide as to what should reasonably be expected from the mill; however, the estimation of the oxide obtained after acid treatment and careful vanning should be near enough, taken at 70 % metal, without going to the extra trouble and expense of smelting, knowing that a further loss in the fire is almost sure to take place. For instance, in the case of the Cornish table previously referred to, where vanning assays gave an expected extraction of 48 %, the table, when installed, actually saved 50 %. It is evident, therefore, that as a guide to mill work the vanning assay has its uses.

THE ORGANIZATION AND EQUIPMENT OF A MINE-RESCUE STATION.

BY J. C. COLDHAM.

THE occurrence of fires in metalliferous mines is fortunately very rare; but, as is well known, the experience of fire-fighting underground is a bitter one. In years past much money has been expended in fighting fires and sealing off affected areas, and, above all, lives have been lost which might have been saved had there been in existence an efficient apparatus by the use of which men could stay for periods in a poisonous atmosphere.

Smoke jackets and helmets, which have been introduced of late years for working in irrespirable atmospheres, have gone far to remedy this deficiency, but great care must be exercised in the selection of an apparatus suitable to the conditions, for it must be remembered that even the best of them have grave faults, which have caused fatal accidents to the wearers even when expert in the use of their jackets.

Thus, it will be admitted that the safety of any selected type will depend on—

(1) The apparatus being in perfect order.

(2) The wearer being thoroughly versed in the use and limitations of his jacket, and confident that he can do a certain maximum quantity of work in a certain time.

In addition to these considerations, the organization of a rescue station (especially a mine-rescue station) must be such that a sudden need will call forth an instant response. This is the fundamental difficulty. The extreme unlikelihood of having to make use of one's training is liable to cause weakening of the will to efficiency, and when the test comes something is lost or broken, or some necessary supply has run short. This lack of

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efficiency is hardly likely to occur in the equipment and management of a central station, where there are men whose sole duty it is to keep up the standard of efficiency; but in the case of a mine running its own station there is the danger of a falling-off, principally because the training and the upkeep will only be of secondary interest in the duties of those attached to the rescue work. It is therefore evident that the main considerations governing the organization of a mine-rescue station may be classed as follow:—

(1) The selection of the jacket is, of course, the primary consideration. One is assisted by the exhaustive tests which have been carried out and the reports of various well-known men, which have established a tentative superiority of certain makes; but it must be remembered that all forms are capable of modification and improvement, so that defects which have occurred in tests may be removed in later forms of the same type of apparatus. Comfort and convenience are necessary. The jacket must be properly balanced on the wearer, and the weight well distributed over his body. The jacket must be perfectly air-tight, and the oxygen supply sufficient for any rate of exertion.

Although the purification of the gases can never be perfect, still it must be certainly established that the purifier of the selected jacket is capable of keeping the CO_2 content of the inspired air within certain limits—a maximum, for instance, of 1 %.

The head and shoulders of the wearer should be clear of all gear, since there will be less risk of damaging the jacket against low backs or projections in the drive.

(2) A sure channel of supply is vital. In the case of a rescue station placed at a great distance from any manufacturing centre, supplies made on the mine from chemicals of which large stocks are kept may either supplement or replace the purchased article.

Thus (*a*) In view of the necessity of having an oxygen compressor to supply high-pressure gas into the jacket cylinders from partially-exhausted storage cylinders, it would be better if a plant to generate oxygen were installed and the compressor arranged to compress

oxygen supplied from the generator, in case the supply from the storage cylinders ran out at a critical time.

(*b*) With regard to the purifier, it would be advisable to have the appliances to prepare sticks or granules on the works.

(3) The selection of the men who are to undergo training, and of those who have to run the station, ranks in importance with the selection of the jackets. If the foreman of the station is not interested in his work, the only satisfactory alternative to procuring a new foreman is to abandon the whole business, especially if the care of the apparatus is only of secondary importance in his daily duties. Supplies must be kept up and jackets must be in perfect order, both from the mechanical and the sanitary standpoint. That is the province of the foreman and his men. Experience has shown that men will become antagonistic to the whole scheme of training unless they feel absolutely sure that the breathing parts of the apparatus are scrupulously clean ; on the other hand, if the mechanism of any jacket is not in perfect order, that jacket becomes a source of danger to the wearer. Everything connected with the jackets must be ready for use immediately the need arises. Upon the despatch with which the jackets are at the scene of action may rest the safety of valuable lives.

The men selected for training, who would probably be selected from amongst the shift bosses and foremen, must be healthy. A rescue team would have to work at the maximum of physical effort under conditions which demand the use of jackets. Thus, taking to account the fact that the best jacket yet manufactured entails of itself a drain on the wearer's energy, it is clear that the man in a team who is not in good health may become a serious hindrance.

Another consideration to be remembered is that men will object to using a jacket after one who is even supposed to have any form of contagious disease. Theoretically, of course, each man should have his own jacket, but this would be practicable only in a central station.

It is vital that the men should take an intelligent interest in the work. Mere willingness, which may be closely connected with cash considerations, is quite inadequate.

At the outset the men must realize that jackets in partially-trained hands are dangerous. The principles of the jacket must be thoroughly understood, and this remark does not apply only to men who have not had a technical education.

The men in training must report what work they have performed at the end of each trial, as they must be carefully watched in order that some idea may be had of their fitness. It has been observed in several cases that the men were always "Physiologically" uncomfortable in their jackets. In these cases it is not worth while continuing their training, as they will never become efficient. If a man is continually fighting for breath his mentality is decreased, as was specially noticeable when training with the older forms of pneumatogen apparatus, where the muscular effort required to draw the gases through the generator was in itself sufficient to keep all one's attention fixed.

A case of the kind came under observation in the trials described below, where a man, usually keen, became quite dull as soon as he put the jacket on. He seemed to have difficulty in breathing, and appeared to be frightened of the jacket. The only work he seemed capable of doing was pulling the "ergmeter," and stated that he felt better when doing this than even walking about. This may be explained by the fact that this work has a similar effect to artificial respiration, thereby assisting the chest muscle, in overcoming the slightly increased resistance due to the bags which was aggravated by the man failing to keep his purifier well shaken up. It seemed that the chest was not capable of overcoming the slight increase of resistance.

The men who will be called upon to use the jackets in emergency should undergo a comprehensive training in their use. This training should be formulated with two ends in view—

(*a*) Absolute confidence in one's jacket is vital. To harness a novice into a jacket and ask him to perform dangerous work in an irrespirable atmosphere is sheer homicide, since even trained men have lost their lives through mechanical defects which could have been easily righted. Therefore the men must thoroughly understand their

own capabilities and the limitations of the apparatus they are using; while the location and setting right of any temporary breakdown should be almost automatic.

(*b*) The men should be under careful observation, and all their performances must be recorded in order that the unfit may be culled out in season.

From the reports of Dr. Haldane and others, coupled with the lessons to be learnt from the too frequent fatalities in connection with the use of pneumatophors, it seems evident that a far more searching selection and training than has hitherto been usual is necessary.

In formulating a system of training, it is necessary to give the men a thorough grasp of the underlying principles and the mechanism of the apparatus. Possible breakdowns and their remedies should be enumerated, and also the ordinary precautions, such as the necessary attention to the purifier in the case of the Fleuss apparatus. In demonstrating to men who have not enjoyed a technical training, it may be better to assume their complete ignorance. In the system of work described below, the practice of working men up to their limit has been purposely avoided. It may be argued that in training for such work a man should do exactly such work in practice that necessity may call for. This is wrong. Urgent necessity is a stimulant which will bring forth the extra effort required. Hard work, however, is a necessity, and kindergarten atmospheres must be avoided as giving a false security.

FLEUSS RESCUE APPARATUS.

Although a description of the Fleuss rescue apparatus. which is used at the Broken Hill Proprietary mine, hardly comes within the scope of this paper, still the jacket and its care has a direct bearing on the organization of the station, and therefore a criticism of its merits and limitations, with a reference to other types, may open up a field for profitable discussion.

Comfort and Adaptability.—The jacket is most convenient as far as distribution of weight is concerned. Everything is tightly fixed to the harness, which consists of a broad belt and braces,

distributing the weight over the hips and shoulders. The cylinders lie low down in the curve of the back, while the placing of the purifier mass in the breathing bag does away with the clumsy regenerating chamber which, in most types, is placed high up on the shoulders, proving a source of trouble, and possibly of danger, when working under a low back. The reducing valve and emergency valves under the left shoulder are rather a trouble, as they are liable to catch in the sides of a narrow drive; but, unless the mechanism is weak. these will not be dangerous. The weight of the apparatus, fully charged, is 37 lb.

Mechanical Construction.—The construction is simple. The circulation of the gases is carried out by the respiratory muscles alone, which, of course, entails an extra effort; but, except in the case previously cited, this has never been noticed by any of the men.

To assist circulation most of the recommended types are provided with injectors, which are undoubtedly necessitated by the added frictional resistance of the separate regenerator and the tubes leading to it. The small resistance of the proto jacket would not justify the use of an injector, as the danger of stoppage of the nozzle and negative pressures, in the case of the injector exhausting the regenerators, must be reckoned with. The South Midland Coal Owners' Mine-Rescue Experimental Committee, 1911, reported two mishaps in trials due to broken screen wires choking the injector. The makers of the Draeger apparatus have recognized this danger, and have supplied a by-pass to the injector in their latest type. Several makers have obviated the second danger by using the injector to force air through the regenerator, but the practice of putting fresh cool oxygen through the hot purifier is not to be recommended.

The reducing valve is set to supply the usual 2 litres per min., which is sufficient for most men, even when at their greatest exertion. In order to test this, however, the men in training were required to run 300 ft. at the rate of about 10 miles per hour, carrying a 50-lb. bag of tailings. The quantity of gas in the bag was noted before and after. It was found that several men—

"C," for instance (see Table III., Schedule 5)—almost exhausted their bags. In order to supply such deficiencies, and also to act as an emergency supply in case of accident to the reducing valve, a by-pass and valve is attached whereby the wearer can let oxygen into the breathing bag at will.

In the trial carried out on 9th June, 1915, details of which are given in Table II., Schedule 5, one of the connections on C.'s bag was leaking, owing to jambing of the connection on the thread; consequently the breathing bag was continually being exhausted, and the trial could be completed only by the use of the by-pass. It will be noticed that the exhaust valve was not used at all. This case also emphasizes the need for a surplus of gas in the cylinder.

The Cooler seems to be rather inefficient. The ratio of cooling surface to content is rather low. It would be better if built of aluminium, with a large central hole, with cotton wool steeped in methylated spirit on both sides of the gas chamber.

The following instance will serve to show the care necessary in preparing the jackets for work. In the trial carried out on 16th June, 1915, the attendant, in saturating the cooler, by some means allowed some of the methylated spirit to get into the gas chamber of the jacket worn by D. Soon after going into the chamber D. began to feel light-headed, and soon afterwards came out with a violent headache, and showing symptoms of being gassed. He was unable to proceed with the test, and complained of the taste of spirit in the inspired air. C., who was superintending the trial, went into the chamber with a little spirit in the cooler. The taste of spirit was immediately noticeable, and soon C. began to feel the symptoms experienced in gassing, and had to come out of the chamber quickly. A gas analysis of the inspired air gave CO_2 3.4 %, oxygen 30 %, showing that neither of these was the cause of the poisoning. A further test showed that by breathing ordinary air across the surface of this brand of methylated spirit similar symptoms were induced.

Face Connection.—As far as respiration is concerned, helmets and masks are better than mouthpieces, as the wearer can breathe

through his nose. Helmets, however, are inconvenient, and restrict the vision so much that they may become a danger, whilst, on the other hand, it would seem that not even an injector would clear the reservoir round the head of impure expired air. Masks and half-masks are so liable to leakage that their advisability is questionable.

Mouthpieces give an almost perfect connection with the lips, if they are properly fastened, although they are not so comfortable as masks. The nose clip is rather uncomfortable, but, if the nose is lightly greased inside, this is not noticeable. The goggles should be specially fitted for each man, since he will be liable to headache if the pull necessary for a gas-tight fit unduly constricts the capillary veins.

The wide, flexible breathing tubes of the proto apparatus are a great point in its favour, the stiff tubes of some makes being very uncomfortable.

Main Supply Valve.—It was found in one case (C., Schedule 3) that, in crawling, the main supply valve was turned off. This was the cause of a fatal accident at the Benwell Colliery, Durham, in June, 1913, where the wearer's valve got partially turned off, and he was asphyxiated in his jacket. This necessitates the use of a clip to keep the valve open.

The design of the valve for an otherwise well-made apparatus is surprisingly bad. To get the valve tight necessitates the use of a spanner.

By-Pass Valve.—In Schedule 6, B.'s by-pass valve became turned on as he was struggling through a narrow drive, blowing up the bag and causing pressure against the lungs. In such a case, where the wearer cannot immediately turn off the valve, he should pull the mouthpiece clear of the teeth, allowing the straps to keep it against the mouth. This allows the air to escape, and the man can breathe. The outflow of air would also keep any bad gas from the mouth till the valve was shut off.

Breathing Bag.—As the bag was liable to become so hot as to be uncomfortable against the stomach, asbestos aprons are fixed behind the bag, which greatly increases the comfort.

Goose-neck.—This is very inconvenient, and catches in narrow places, especially when the wearer is crawling.

SURFACE EQUIPMENT.

The surface equipment is directly under the control of the fire-station foreman, who is, in respect to the rescue station, under the supervision of the underground department. The staff of the fire station is required to keep the jackets and subsidiary plant in order, to keep the stock of supplies up, and to prepare everything for the trials.

The main equipment consists of four Fleuss jackets, which are kept in the ambulance room on special stands. The face connections, breathing bag, etc., are kept in a dust-proof drawer under the stand, and the whole jacket has a canvas cover over the stand.

Great care must be given to the cleaning of the apparatus: the metallic connections must be kept free from rust and verdigris, whilst the inside of the breathing parts must be thoroughly disinfected after use, and all the rubber parts are stored in French-chalk. An air jet is provided to clear it of all traces of dust, which would cause coughing if drawn into the trachea. The parts of each jacket are numbered, and must be assembled together, so that any faults in the jackets may be located.

The small oxygen cylinders are charged from 15 large 200 cub. ft. storage cylinders, which are kept in the store. These small cylinders are charged to 100 atmospheres in the longer trials.

In order to get the maximum quantity of oxygen from the storage cylinders without using the booster pump, the small cylinders are filled in two stages from the storage cylinders. First of all the gas is run in from a bomb, which has been partially exhausted, and the pressure is brought up to the 100 atmospheres necessary for the trials from another cylinder. When the storage cylinders are exhausted to 75 atmospheres they are turned over to the engineering department for use with the oxy-acetylene welding apparatus. The method of booking the quantity of gas used is shown in Appendix 2, from an actual entry in the gas

ledger kept in the fire station. By this means it is possible to get eight refills of 100 atmospheres from each storage cylinder.

In order to get the higher pressures necessary for serious work, a boosting pump is installed to compress the gas from partially depleted cylinders. This is an ordinary Siebe Gorman compressor designed to draw from atmospheric pressure and has been adapted to pump from the storage cylinders. A water-lubrication system was installed, as it was found that the piston-packing and valves would not keep tight when working dry.

About 20 lb. of caustic soda, in sealed tins, is kept in stock in case of emergency, but the sticks for ordinary work are cast at the assay office by the fire-station men. The melting-pot holds about 12 lb. of soda, and must be kept tightly covered to prevent CO_2 getting into it. For this purpose the down-draught of an open assay furnace is good. The moulds hold 12 sticks, 6 in. long x 3/16 in. diam. These moulds are split and clamped, so that the sticks can be easily freed. The soda sticks assay about 94 % NaOH.

Pulmotor.—The station is supplied with a pulmotor and its own special oxygen cylinders, which can be carried to the scene of action, if necessary.

Auxiliary Jackets.—In addition to the four proto jackets, the mine has in stock twelve pneumatogen jackets of the old single-cartridge type. These jackets are unreliable in any hard work, but in certain cases, which will be explained later, they will be of use.

These jackets are kept in separate pigeon-holes in a dust-tight cupboard, with a fresh regenerator cartridge ready in front of each jacket.

Hand Lamps.—Four electric hand lamps, giving a light of 3 c.p. for 6 hours, are kept ready. These are periodically tried, and, to prevent the cells being exhausted by casual use in the station, they are kept in a flimsy box, which can be easily broken open in case the lamps are suddenly needed. The key to this box is in the possession of the electrician, and a duplicate is kept in the underground office.

Testing Valves.—In order that the gas supply to the breathing bags may be periodically tested, a displacement meter has been devised, which has proved satisfactory (fig. 1). It is an ordinary 1-litre flask (A), which moves freely within the guides (B) of the water reservoir (C). This flask is balanced by a small tin buoy to float freely to the 1-litre mark. The inlet tube (D) is carried to the top of the flask when this is submerged. The reducing valve is coupled to the end of this inlet tube at E, and the displacement of water by the gas timed. In order to quickly refill the flask, the inlet tube is connected with an injector (F) having the necessary valves (GG).

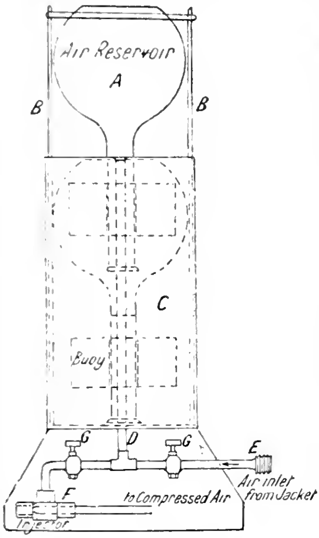


FIG 1.
SKETCH OF AIR-METER.

Air-tightness.—The apparatus for this is merely a long water-gauge, which can be fixed to the seat of the exhaust valve. The bag can be inflated to about 10 in. water-gauge, and the air-tightness gauged by the length of time the gauge will hold up.

Supplies.—The necessary stocks of soda, sulphur, spare parts, etc., are kept in readiness at the fire station.

TRIALS.

The system of training the men is shown in Appendix 1. Schedule 1 is taken up in demonstration of the principles of the jacket, but it must be understood that such demonstration was not confined to the lesson, but was kept up throughout the trials. Schedule 2 was carried out in the ordinary up-cast atmosphere, the idea being to accustom the men to moving about in the jacket and to taking the necessary precautions for ensuring a good gas supply. The next three tests were carried through in the gas chamber in a dense atmosphere of SO_2, which ensured

that even the smallest leak in the respiration system would cause the man to pocket his pride in leaving the chamber rather than brave the evil in order not to seem inferior to his mates.

It may be stated that in the few cases where the nose clips were knocked off the men invariably stayed in the chamber while re-adjusting them.

Schedule No. 6 was meant to test the men's staying power. The air was very hot, and the condition of the drive was awkward. It took 40 minutes in the first trial to get the dummy out, and in doing this the men were extended to their utmost. The goggles obscure the vision a good deal, and bumps are frequent. It is very doubtful whether the men could go for another 15 minutes at the same rate. The temperatures and the pulse readings in Tables II. and IV. show this very clearly. The test turned out much more trying than was expected, and afterwards even the exercise of walking quickly round the drives was tiring.

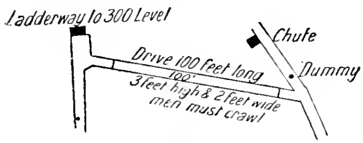


Fig. 2.
Sketch to Explain No. 6 Schedule.

It will be noticed that, although a fully charged Fleuss jacket is designed to last for two hours, it was not deemed expedient to accustom the men to more than $1\frac{1}{2}$ hours work, for several reasons—

(1) These trials and various tests carried out in other fields show that for periods longer than $1\frac{1}{2}$ hours the temperature of the inspired air is liable to cause inconvenience, and the purifier may become so hot as to be painful.

(2) It was considered advisable that men should not be accustomed to working near to the exhaustion point of the jacket. If a man knows his capabilities only up to $1\frac{1}{2}$ hours, it is only some unavoidable delay which will prevent his being out of the danger zone by that time, whereas if he works, through custom, to within a few minutes of the exhaustion point of the jacket, any delay may prove fatal. The extra half-hour's supply is meant to meet such a need.

(3) In a hot, moist atmosphere such as would be met with, a time limit to effective effort is imposed by the rapid development of fatigue, particularly mental fatigue, which is, of course, not peculiar to work in rescue apparatus. After a man has been doing hard work continuously for some time with the wet-bulb temperature over 83° he will be overcome by a strong feeling of lassitude, and will become particularly indisposed to mental effort. In such atmospheres, even with frequent rests, $1\frac{1}{2}$ hours would probably be too long.

The gas chamber consists simply of a disused drive in an up-cast section sealed off at each end. This is 70 ft. long and 8 ft. x 8 ft. in section. There is a gas-tight door in each stopping, and an observation window at the working end. Electric lights are installed to adumbrate the gloom, and an ergmeter is placed at the observation end. This is simply a 40-lb. weight, which can be lifted 5 ft. to a pulley, over which the pulley-rope passes. The material for the trials is stacked along the side of the chamber. The temperature of the chamber was nearly constant at 76° dry bulb, 74° wet bulb. The fumes were kept uncomfortably dense, so that the men could not do so much as in a clear atmosphere; but Table II. shows from the pulse readings that the schedules were conscientiously carried out—indeed, most of the men did more than was asked of them. One of the fire-station men was in attendance at the trials, and at the later trials a member of the assay office staff made analyses of the inspired air.

In Table II. it will be seen that three pulse readings were taken —one before the trial, one immediately after, and the third 15 minutes after—to gauge power of recovery of the various men. In the trials a number is given to each man, and Table I. shows their ages and measurements.

The procedure of the trials was, broadly, as follows:—

Each man saw to the assembling of his own jacket, examining the valves, and noticing whether the connections all had washers. The bodily temperature and pulse of each man is read and recorded. The pressure in the cylinder and the time were

noted just before each man started. On finishing the trial the pressure and time were again recorded. The main valve was then turned off and the nose-clip removed, the man placing his tongue in the mouthpiece until the gas sample is drawn off. The fullness of the bag is noted, and a rough idea obtained of the number of times the relief valve was used. The bodily temperature and pulse were immediately taken, also the temperature of the inspired air and the temperature between the bag and the shirt. The man is required to sit down quietly for 15 minutes, when the pulse is again taken. The soda is examined and weighed before being thrown out.

The figures from the latter trials are shown in Tables II. and III.

At the beginning of the training several of the men formed the habit of throttling the air supply when not working hard enough to use the supply. This is bad practice, and was discouraged. In the first place, a man might forget to turn the valve back again before starting to work hard, and, secondly, the flushing of the bag is a good thing, as it purifies the gas and tends to cool it.

Only one man was rejected as unfit. In going through Schedule 2, G. did not seem to be at home. In Schedule 3, after remaining about 10 minutes in the gas chamber, he came out complaining that he could not breathe, and had experienced a choking sensation in the chest. His pulse was hardly affected, but he was perspiring freely, although he had not started to work. It was quite apparent that he was afraid of his jacket, and, although he would have got over this in time, he was rejected as being unfit to cope with an emergency.

An examination of Tables I. and II. shows that, roughly speaking, the older men do not recover so rapidly from the effects of exertion. On the other hand, although his pulse recovery was slow, B., who is small and compactly built, was one of the best men tried. This was due to a great extent to his personal habits, but it was also largely accounted for by the fact that he was what may be termed a light breather. This was shown very clearly in trying the pneumatogen jackets, as B. could use his jacket comfortably

longer than anyone else, owing to the fact that the regenerator was not being so heavily taxed. With the proto jackets he was continually using the relief valve even when working hard. F. also used less gas than the others.

It seems to be a rule that a man who uses up a small quantity of oxygen in respiration is preferable to one who is a heavy breather. With the light breather the purifier gets less work to do, and also a good surplus of gas, which has to be exhausted. This is undoubtedly an advantage. In the first place, the temperature of the inspired air is kept down, while every time the exhaust valve is used some of the residual nitrogen is let off, so that even when extreme exertion exhausts the bag the oxygen content of the inspired air will be high.

In Table III., Schedule 5, D.'s bag, when he came out of the chamber, was almost exhausted, and the dangerously low oxygen content of the inspired air was due to his not having exhausted the bag once during the trial. If the inspired air becomes hot the system can be appreciably cooled, provided there is sufficient in the cylinders, by opening the by-pass and relief valve together for a short while.

Purifier.—The purifier, on the whole, was satisfactory.

A test (Appendix 3) shows that potash will take up a far larger quantity of water and give out less heat in absorption. Soda, on the other hand, in proportion to its weight, will take up more CO_2. Its chief advantage, however, is that it remains more open than potash, which becomes so pasty that the absorbing surface is seriously diminished. When starting out, the wearer should go slowly for a few minutes. Very dry, pure soda is liable to be slow in action until it becomes wetted, and in a trial C.'s inhaled air after 3 minutes contained 3.4 % CO_2, which is rather too high. In this case C. started hard work immediately on turning on the supply valve.

The importance of thoroughly shaking up the purifier cannot be stressed too much. Several cases will serve to emphasize this :—

(1) In Table III., Schedule 5, B.'s air became hot after 45 minutes' work brattice-building. After thoroughly kneading the purifier between the fingers the air again became comfortable in about 10 minutes.

(2) In the same schedule D.'s air was hot at the end of the test, and the CO_2 content, .8 %, was the only notable CO_2 reading recorded during the tests. On examination it was found that the purifier had caked into three large lumps, which would easily explain a high CO_2 content. The last schedule showed about the limit of the purifier as far as comfort is concerned. In nearly every case the inspired air at the end was hot on the throat, Several of the temperatures taken behind the bag were high, but the men did not seem to suffer any inconvenience. The asbestos aprons seem to be necessary, as several of the bags were quite hot when the apron was drawn aside.

FUTURE WORK.

With regard to the completion of the training, another six men will be taken in hand as soon as possible.

It is considered that twelve men is all that can be handled effectively at the mine, and this number should suffice to cope with any emergency which may arise.

No attempt will be made to train the miners, on account of their liability to leave the mine.

In order to keep the men fresh to the use of the jackets, trials will be undertaken at three-monthly intervals. These will be of 1½ hours duration, the first half-hour of which will be in the smoke chamber.

POSSIBLE PROCEDURE IN CASE OF FIRE.

Although the procedure in case of fire cannot be foreseen with any certainty, nevertheless, in order that the organization of the station may be carried out to its logical conclusion, there must be some rough prevision as to the movement of rescue parties, etc.

In the first place, it is necessary to have a thorough knowledge of the ventilation currents of the mine. This matter is simple where artificial ventilation is installed. At the Proprietary mine a ventilation plan is kept, which is brought up to date yearly, or as new connections make radical changes.

Each team should have a recognized leader; this was provided for in the trials. Certain men had to lead all the work, and these men would be in charge under actual working conditions.

RESCUE WORK.

In rescue work the procedure would depend entirely on the conditions. In all probability the men would work in one team of four.

The ambulance men would be ready at the nearest point of safety. If it were likely that men would be brought back through gas, a pneumatogen jacket or two would be taken for the purpose.

In any work it would be advisable for parties to carry tools, as these may prove necessary, and may be left behind if they become a hindrance.

FIRE FIGHTING AND SEALING OFF.

Here the procedure can be foreseen more clearly, for when a fire breaks out it is quite probable that if a trained man is at hand and the apparatus is ready, hoses may be taken forward to the seat of the trouble and the fire extinguished before it is too late. If, however, jackets were not available, there is no man who could face the smoke and hot gases which would be given off.

In sealing off a fire area the jackets would be invaluable for carrying forward brattices, building permanent stoppings, and general reconnaisance work. The procedure in building a permanent stopping would be somewhat as follows:—

The temporary stopping would in all probability not be necessary.

One pair of men would proceed to the spot where the stopping was to be built and commence to prepare a foundation, while supplies were being brought forward to the nearest point of safety.

If the stopping were not too far from this base of supplies the material would be taken forward to the work by men wearing

pneumatogen jackets. If the distance were too great for the pneumatogen jackets to be used with safety, the second pair of proto apparatus would be used for this purpose.

Should pneumatogen jackets be used for supplying material, the building gang would probably consist of three men, the fourth jacket being kept in reserve. The men of the building gang would return at different times to the base to have their cylinders re-charged or to be relieved.

CONCLUSION.

Although the present system of training is not so long as that carried out at certain of the large stations of England and the United States of America, still, such as it is, the trials showed that the men were able to work at high pressure under trying conditions, and without the call of necessity to keep them going.

The two main objects of the trials have, therefore, been achieved. Firstly, all the men showed themselves capable of doing any work they are likely to be called on to carry out; and secondly, both those responsible for the training and the men themselves, while realizing the limitations of the jackets, are confident that they can meet any emergency which may arise.

In conclusion, since prevention is better than cure, it may be hoped that, owing to the precautionary measures taken against fire underground at the Proprietary mine, no need for the jackets will ever arise.

Table I.

Showing Physical Condition of the Men Trained.

Name.	Age.	Height.	Weight.	Chest. Contracted.	Chest. Expanded.
		ft. in.	st. lb.		
A.	44	5 8½	11 10	36½	38½
B.	58	5 3	9 8	37	40
C.	31	5 11½	12 10	37	41
D.	37	5 7	11 0	37½	40½
E.	53	—	9 5	—	—
F.	33	5 10	11 5	35	39

TABLE II.

SHOWING TEMPERATURE AND PULSE CONDITIONS DURING LATER TESTS.

Name.	Date.	Schedule.	Time in Jacket.	Temperature.		Pulse.			Remarks.
				Before.	After.	Before.	After.		
							Immediately.	20 min.	
A.	4/2/15	4	75	—	—	92	105	—	
	9/6/15	5	90	98.5	not taken	84	112	94	
	7/7/15	6	90	99.3	100.2	78	124	92	
B.	2/6/15	4	75	98.3	98.6	82	124	92	
	16/6/15	5	90	98.4	98.5	84	120	88	Got caught in the goose-neck in crawling; he had to struggle to get out.
	30/6/15	6	105	98.1	not taken	80	140	104	By-pass got turned on, causing inconvenience, but kept mouthpiece in.
C.	4/2/15	4	75	—	—	84	124	—	Nose clip was too tight, causing headache; took off in gas and adjusted.

Table II.—*Continued.*

	9/6/15	5	75	98.0	not taken	86	128	96	Gas supply ran short through leaky joint.
	30/6/15	6	90	98.2	99.4	84	112	92	Got badly caught by stone slipping from side and jamming, while goose-neck was caught across water-pipe.
D.	2/6/15	4	60	98.2	99.8	78	140	100	Got lungs full of gas through inhaling past loose mouthpiece, 45 min. after start.
	23/6/15	5	75	98.4	98.5	80	132	92	Gas supply ran out.
	30/6/15	6	95	98.3	not taken	78	116	100	
E.	25/5/15	4	75	98.4	98.5	78	96	80	Oxygen pipe became unscrewed through careless adjustment; fixed immediately, but he felt bad for some time.
	23/6/15	5	90	98.3	99.0	78	132	104	
	—	6	90	98.4	101	78	128	104	
F.	25/5/15	4	95	98.4	98.6	80	120	82	
	23/6/15	5	90	98.4	98.5	78	98	80	
	—	6	95	98.8	100.6	74	128	94	

TABLE III.

SHOWING EFFICIENCY OF PURIFIER AND TEMPERATURE OF INSPIRED AIR AND BEHIND BREATHING BAG AT THE END OF THE TWO LAST TRIALS.

Schedule 5.—

Name.	Date.	Time in Jacket, min.	Amount of Gas in Bag at end.	Condition of Purifier.	Number of Times Bag Exhausted.	Temp. after Trial.		Gas Analysis, Inspired Air.		Remarks.
						Inspired Air.	Behind Bag.	CO_2 %	O %	
A.	9/6/15	90	Medium	Good; some lumps showing	6 or more	98.6	107	.1	30.6	Inspired air warm at end.
B.	16/6/15	90	Full	Good	Continually	93.2	124	.1	80	Inspired air became hot during trial, but on shaking bag well it cooled down.
C.	9/6/15	75	Low	Good and well separate	None	98.6	113	.1	21.9	Leak in joint of jacket, caused running down of gas supply.
D.	23/6/15	75	Almost empty	Very bad; the soda was in 3 hard lumps	1 or 2	93.0	104	.8	18.8	Low O, due to emptiness of bag; high CO_2, due to bad purifier.
E.	23/6/15	90	Half full	Badly clotted	About 12 times	98.5	98	.1	45	Breathing warm at end.
F.	23/6/15	100	Full	Very good and open	Frequently	98.0	104	.1	70	

TABLE III. - *Continued.*

Schedule 6.—

Name.	Date.	Time in Jacket, min.	Amount of Gas in Bag at end.	Condition of Purifier.	Number of Times Bag Exhausted.	Temp. after Trial. Inspired Air.	Temp. after Trial. Behind Bag.	Gas Analysis, Inspired Air. CO_2 %.	Gas Analysis, Inspired Air. O %.	Remarks.
A.	7/7/15	90	Medium	Very good and open	Frequently	101.5	104	.2	60	
B.	30/6/15	105	Low	Bad, lumpy	Frequently	104	114	.1	36.6	Inspired air quite comfortable.
C.	30/6/15	90	Low	Fair ; a few lumps	3 or 4 times	106	95	.2	23.2	Inspired air dry and hot at back of throat.
D.	30/6/15	95	Low	Good and open	12 times, about	101	98	.1	31.4	Inspired air warm.
E.	7/7/15	90	Medium	Good	Frequently	101.6	106	.1	86	Air cool throughout
F.	7/7/15	95	Full	Clogged on one side	Frequently	100.2	106	Not taken		Air warm on throat. Clogged side of bag noticeably warmer than other

Table IV.

Readings, Schedule 6.

(After bringing out dummy—about 36 minutes).

Name.	Temperature.	Pulse.	Remarks.
A.	100.4	136	Taken 5 min. after stopping
B.	100.8	—	,, ,, ,, ,,
C.	101	140	Taken immediately.
F.	101	140	,, ,,

Temp. of level, 82° dry bulb ; 80° wet bulb.

Appendix I.

Course of Trials in the Use of the "Proto" Jacket.

Schedule 1.—

(*a*) Instruction in the use, mechanism, and principles of the jacket.

(*b*) Possible accidents and their remedies.

(*c*) Ordinary precautions necessary to ensure a good gas flow.

(*d*) Piecing the jacket together and dressing.

(*e*) Walking round the room breathing from the jacket.

Schedule 2.—45 minutes using the jacket as in gas—temp. 74° dry, 73° wet.

(*a*) Walk 180 ft. along crosscut and back, carrying piece of rail weighing 160 lb.—400-ft. level, Weatherley's crosscut.

(*b*) Crawl 60 ft. to ladderway.

(*c*) Climb down ladder to 640-ft. level. The manholes in this ladderway are very small, and will show the wearer the capabilities of his jacket.

(*d*) Walk to the belt conveyor and crawl 60 ft. along the conveyor.

(*e*) Walk along level to finishing point carrying 8 ft. 10 x 10 between two men for 50 ft.

(*f*) Finish up time building a dry wall from the stack of stones at the finishing point.

Schedule 3.—1 hour work in fumes in gas chamber—temp. 96° dry, 72° wet.

(*a*) Sit down for a few minutes, then walk quietly round, getting used to the conditions.

(*b*) Work ergmeter 20 to 40 times.

(*c*) Crawl length of the chamber and back. Try crawling flat on stomach for short distances.

(*d*) Saw two 5 x 5 props. Carry bricks from stack and place in a dry wall.

(*e*) Carry 50 lb. bag of sand up and down chamber six times.

(*f*) Alternately walk up and down chamber and work ergmeter.

(*g*) Pull weight hard 50 times immediately before leaving chamber.

In all, the ergmeter should be worked 150–180 pulls.

Schedule 4.—$1\frac{1}{4}$ hours in gas chamber.

(*a*) Walk 12 laps of the chamber.

(*b*) About 30 blows each man, boring a hole in side of the chamber.

(*c*) Get 6 props from stack; place along the chamber at 6-ft. intervals for bratticing. These props are 9 ft. long, and must be cut.

(*d*) Work ergmeter 20 or 30 times, then take a second spell at hammer and drill work.

(*e*) Carry 50 lb. bag six laps of chamber.

(*f*) Take down brattice props.

Finish up time alternately walking about and working ergmeter; take 30 pulls just before coming out of the chamber.

Schedule 5.—1½ hours in gas chamber.

(*a*) Walk 12 lengths of the chamber.

(*b*) Work hammer and drill 30 passes to each man.

(*c*) Put up brattice similar to Schedule 4; connect 30 ft. 1-in. pipe, and secure along the top of the line of props.

(*d*) Carry 50 lb. bag several lengths of the chamber at a smart pace.

(*e*) Pull down brattice line.

(*f*) Bolt together five 6-ft. lengths of 11-in. ventilating, four bolts in each; take apart again and stack in place.

(*g*) Transfer the brick stack, carrying six bricks at a time.

(*h*) Work ergmeter 20 times at intervals during the test.

Schedule 6.—1½ hours in levels, three in team.

(*a*) Two men on 300-ft. level, Darling shaft, climb to 250-ft. level, fixing pulley in the shaft, and haul up stretcher, canvas, and tools fixed to rope by No. 3, who then joins his mates at the 250-ft. level. Take stretcher along the cross-cut as far as possible, and then crawl along the level 50 ft., carrying tools and canvas to the mullock chute (see sketch, Fig. 2). Bring back dummy along the level, place on the stretcher, carry to the shaft, and lower to the 300-ft. level, where he is taken off by one of the men, who has climbed down. Rest 10 minutes with mouthpieces out, and then finish up the time by investigating the workings of Block 13, moving at about 4 miles per hour.

The 250-ft. level is broken into and the floor is covered with broken quartzite. There is a water-pipe about 2 ft. from the floor of the drive under which one must crawl, as the drive is too narrow to keep to the side of it. The temp. is 82° dry, 80° wet.

Appendix II.

Showing Use of Oxygen in Rescue Jackets.

Date.	Jacket Cylinders.			Storage Cylinders.						Remarks.
				Low Pressure Cylinder.			High Pressure Cylinder.			
	No. Filled.	Press.	Amount Air, Cubic Feet.	Distinguishing Number.	Pressure. Before.	Pressure. After.	Distinguishing Number.	Pressure. Before.	Pressure. After.	
16/6/15	1	105	—	—	—	—	—	—	—	
	1	100	—	—	—	—	—	—	—	
	1	75	$23\frac{1}{4}$	—	—	—	20056	115	105	
23/6/15	3	100	$24\frac{3}{4}$	20056	105	95	20055	125	115	20056 to fitting shop
30/6/15	3	100	$24\frac{3}{4}$	—	—	—	20055	115	100	

The quantity of oxygen used from any cylinder is adjusted from its pressure when handed over to **engineering department.**

Appendix III.

Experiments showing the rate of solution of a given weight of caustic soda (NaOH), and caustic potash (KOH), in a given volume of water, and the rise of temperature caused thereby.

(1) 13.1715 grm. NaOH dissolved in 50 cc of water in about 4½ minutes, the temperature rose from 60° F to 146° F, or 86° F.

1 grm. of NaOH will give a rise of 6.52° F.

(2) 13.7 grm. of KOH dissolved in 50 cc of water in about 1½ minutes, the temperature rose from 60.5° F to 114° F, or 53.5° F.

1 grm. of KOH will give a rise of 3.9° F.

THE SCHEELITE-GOLD MINES OF OTAGO, NEW ZEALAND.

By C. W. Gudgeon.

The scheelite-gold deposits of Otago occur payable in only two known districts—Macrae's Flat (Waihemo County) and Glenorchy (Lake County) (Fig. 1). Broadly speaking, there are two main systems governing the metallurgical treatment, which are —(1) those running primarily for a scheelite product, gold as secondary, with coarse crushing; and (2) those giving preference to gold-saving, scheelite as a by-product, fine crushing. Mines operating in the Glenorchy district are primarily scheelite savers, while those of the Macrae's Flat district aim for a high gold extraction.

Fig. 1.
Province of Otago, New Zealand.
Scale—100 miles to 1 in.

Of the numerous mines that have been exploited past the primary stage, only three have ever reached the stage of constant and steady production, and, strangely enough, these three are, as far as treatment is concerned, opposed to one another. These three mines are the Glenorchy mine of that district, and the Golden Point and the Highlay mine of the Macrae's Flat district. Many other propositions have at different times shown promise, but, owing to their not being able to obtain a constant production, due to the patchy deposition of the scheelite, have been compelled to close down.

[This paper is separately bound, and may be if so desired, detached complete from this number.]

The first attempt to establish the scheelite industry in Otago was made in 1875–1880 (?) by a company directed by the late Professor Ulrich, of the Otago University School of Mines, on the property now owned and worked by the Glenorchy Scheelite Mining Co. Ltd. The operations were somewhat crude; coarse crushing through rock-breaker, sizing, and hand-picking formed the treatment. The low price for scheelite then offering closed the mine, and it lay idle until 1905, when the present company took it up and brought it to a successful issue.

The next attempt was made by the Donaldson Bros., who then owned the Golden Point mine, and who had been working it in a small way from 1891 as a gold proposition. Being advised by Arthur Kitchener, a visiting (London) mining engineer, a trial shipment was made to Germany, which, resulting in a net profit of £15 per ton, caused attention to be given to the saving of the scheelite concentrates (which up to this time had been a constant source of trouble to the millmen in silting up on the amalgamation plates), and from this time forward the Golden Point mine and others extended their saving appliances for the scheelite, until in the year 1905 the industry (due, in great measure, to the then high price ruling for the ore) became firmly fixed in the Otago province.

The following descriptions show the practice obtaining on the two main mines in 1914, and the Highlay mine in 1913, when it ceased operations:—

Golden Point Mine.*

Macrae's Flat, situate 50 miles north of Dunedin (Figs. 1 and 2), is on a high plateau at an elevation of 2000 ft., described by Professor Park as a "block mountain"; country-rock mica-schist, lying horizontal; classified by Hutton as Silurian, and later by Dr. Marshall as Maitai (Jurassic–Triassic).

A mineralized belt or zone runs more or less continuously from the Stoneburn mine to the Highlay mine, a distance of 16 miles,

* Quoting largely from "Scheelite Mining in New Zealand," by C. W. Gudgeon, *Aust. Mining Standard*, 13/11/1913.

right across the plateau. This zone is ½ to 2 miles wide, with occasional cross belts, is highly mineralized, and carries isolated chutes of ore; in places the outcropping is visible for stretches of 3 miles.

The Golden Point reef systems, which are worked by adit levels, are three parallel lodes, running conformable with the altered zone strata, striking N. 40° W., and dipping 10–25° N.E.,

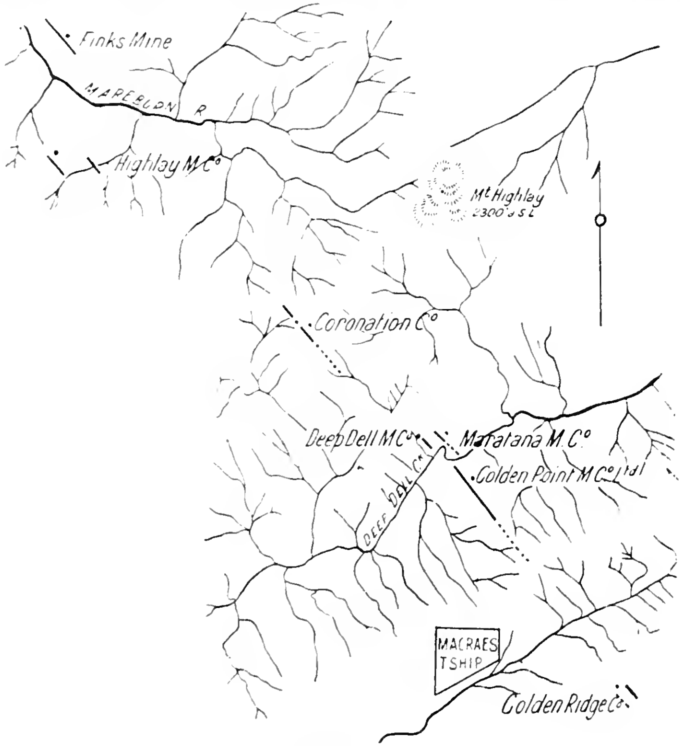


FIG. 2.

MACRAE'S FLAT REEF SYSTEM.

Ore-bodies Opened ——— Outcrops ....

Scale—160 chains to 1 in.

separated 15 and 25 ft. respectively. The top lode, No. 1, is 3 to 4 ft. in thickness, and carries very low values in gold and scheelite (with occasional small pockets of rich ore): No. 2 (centre) is 3 to 6 ft. in thickness, and carries ½ to 1 oz. gold to

the ton, and is highly impregnated with scheelite, and also holds many pockets of almost pure scheelite. No. 3 is small, and of no importance. Going south, these three lodes appear to junction into one main body before petering out, and northwards, towards Deepdell Creek, spread out from each other before dying out. The gangue is a quartz one, and has associated with it in the ore-bodies—Au, $CaWO_4$, FeS_2, As, S, and $MnOSiO_2$.

The lodes are very much faulted (step and reverse faults predominate), and in consequence are somewhat hard to follow, requiring careful timbering. In the old days the lode was opened up by adits along the lode. but this system, while having the advantage of prospecting the ore-body being followed, has been replaced by drifts under the lodes, cross-cutting, and vertical rises, which has enabled the ore-body to be kept under better control, and also cut down the cost of ore getting. Owing to the soft nature of the schists, levels 6 x $4\frac{1}{2}$ ft. are driven on day wages (10s.) as low as 6s. per foot.

Stopes are timbered with ordinary king post, cap, and struddles, with spaced lagging, kept where convenient to the size of the lode being extracted. Stoping operations consist in taking "slicing" cuts working towards home, no filling being run in; pillars of ore are left round the ore-pass, then removed just before the section is abandoned.

The occurrence of scheelite in the lode is very patchy. Pockets of ore being the exception rather than the rule, are now only worked out in the ordinary course of stoping operations; it is seldom, however, that there is no scheelite present. As a general rule the pockets of high-grade scheelite occur adjacent to the walls, deposition on the footwall being the more frequent. The most favourable indication of a pocket of rich ore is a footwall of hard, black, slickensided schist, very highly polished and mineralized; with the lately-obtained good pockets of scheelite this indicator has been always present.

The treatment at the mine is to bag all firsts and send to the surface, seconds being sent out with ordinary crush dirt to the mill. The firsts are then hand-knapped up to 60–65 % WO_3 and bagged for shipment, rejects being passed to mill for treatment.

The costs on this mine are moderately low; during the last quarter of 1914, on 1998 tons treated, they were:— .

COST SHEET.

	s. d.	s. d.
MINE—		
Supervision	0 5.6	
Stoping	2 7.5	
Development	2 0.7	
Tramming	2 2.1	
Repairs	0 4.9	
		7 8.8
MILLING—		
Breaker, jigs	0 1.9	
Milling	2 3.9	
Concentrating	0 4.0	
General	0 3.8	
KCN plant	1 1.3	
Magnetic plant	1 6.0	
		5 8.9
Administration and general expenses		1 2.8
Gross total		14 8.5

1000 tons tails cyanided at 2s. 2.5d.
32 tons pyritic scheelite treated at £4 16s.

MILL.

The milling operations for clean non-pyritic ores are simply crushing to 30 mesh, amalgamation, then concentrating out of the scheelite (which, owing to its high specific gravity, presents no concentrating difficulty in separating from the gangue), ordinary percolation KCN treatment of the sand tails, and storage of the slime tails. Of late years, however, with the increasing depth of workings (500 ft.), the lodes have passed into the sulphide zone, in consequence of which the scheelite is associated with the pyrites, and they are saved together as a low-grade concentrate. The present treatment is illustrated by flow sheet (Fig. 3).

The plant consists of a 55 b.h.p. Cambridge suction gas plant, breaker 10 x 7 in., 10 head (900 lb.) mill, feeders, Wilfley tables, raff wheel, power jig, 250 ton KCN plant, 6-ft. cylindrical drying furnace, 9-ft. cylindrical roasting furnace, and Weatherill double-

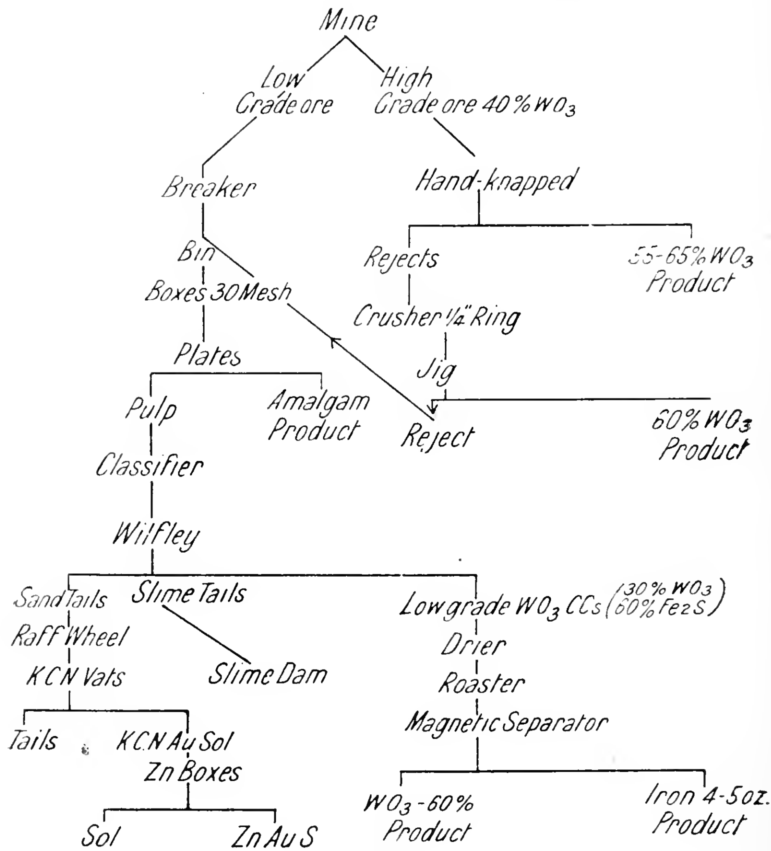


FIG. 3.
GOLDEN POINT MINING CO. LTD., FLOW SHEET, 1914.

roll magnetic separator, U.S. type 2. The dynamo that generates the current for the separator also lights the mill. The plant is suitable for ore treatment from either the oxide or sulphide zones. The mill has a capacity of 30 tons per day through a 30-mesh.

Of the gold, which averages 12s. per ton, 46 % is caught on the plates, 36 % in KCN plant, 6 % is in tails reject, 8 % to slimes dam, and pyritic gold saved in magnetic treatment is 7 %. The average low-grade concentrates contain $CaWO_4$ 30–40 %, pyrites 50–55 %, and SiO_2 10 %, with a gold content of $2\frac{1}{2}$ oz., and runs from $\frac{1}{2}$ to 1 % of the ore milled.

Complete analysis of a sample of the low-grade pyritic scheelite is as under :—

	%
WO_3	36.1
SiO_2	10.8
As	9.5
Fe	7.7
S	4.4
CaO	9.3
Fe_2O_3	22.2
	100.0

which, for purposes of magnetic treatment, are grouped into three heads—

	%
$CaWO_4$	45.4
Iron	43.8
SiO_2	10.8

The magnetic separation, as carried out, consists in thoroughly drying the concentrates, then roasting the pyrites to the magnetic sulphide stage, screening, and separation.

In roasting, using a cylindrical furnace (Fig. 4), it was found that by elevating the calcined concentrates and allowing them to drop 8 ft., with free access to the air, their nature became very pronounced. In magnetic separation (by deflection, Fig. 5), 15 % of the $CaWO_4$ and 21 % of the SiO_2 present is deflected over with the magnetic iron, while running on high amperage. This iron product is submitted to a re-treatment and re-roast, resulting in a clean iron product containing 5–6 oz. gold to the ton, and $CaWO_4$ 73.4 %, equal to approximately 61.2 % WO_3. Concentrates that may be high in SiO_2, before going to the separation

plant, are jigged (Hutch product) to bring SiO_2 contents down to 10 % or under, which is the pre-magnetic standard adopted.

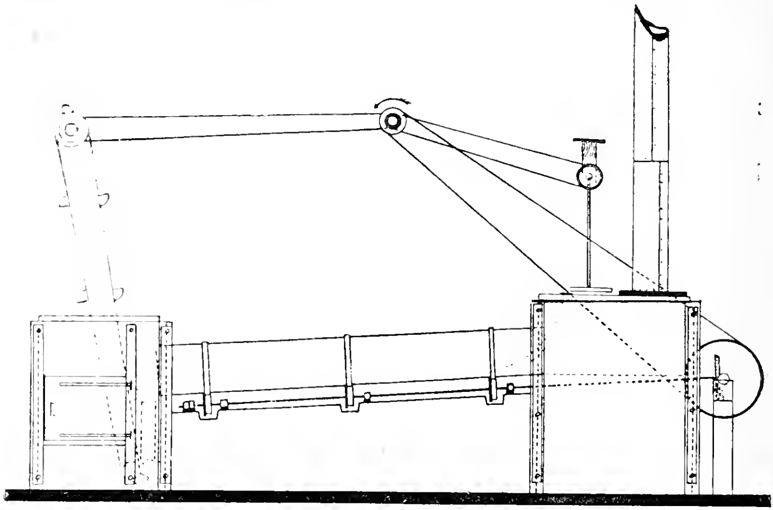


Fig. 4.
Cylindrical outside-driven Roasting Furnace for Pyritic Scheelite Concentrates.
Scale—¼ in. to 1 ft.

It is commonly stated on this field that whenever the scheelite values are high there is a corresponding drop in the gold values, and vice versa; and the graph (Fig. 6), compiled from reliable records (recoveries only), bears this statement out, as in every case but one (1911–12) the respective movements of the values in relation to each other have been apart. This statement must not be constructed to mean, however, that a high presence of scheelite in a face implies always a low value of gold, as 50 % (WO_3) specimens have been obtained with adhering particles of gold which have assayed as high as ½ oz.

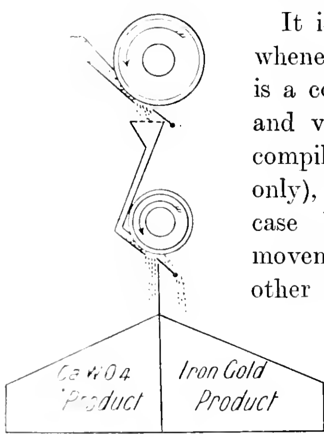


Fig. 5.
Golden Point Mining Co. Ltd. Magnetic Separator. Diagram Illustrating Treatment.

to the ton, which is considerably above the average gold recovery covered by the graph of 12s. per ton.

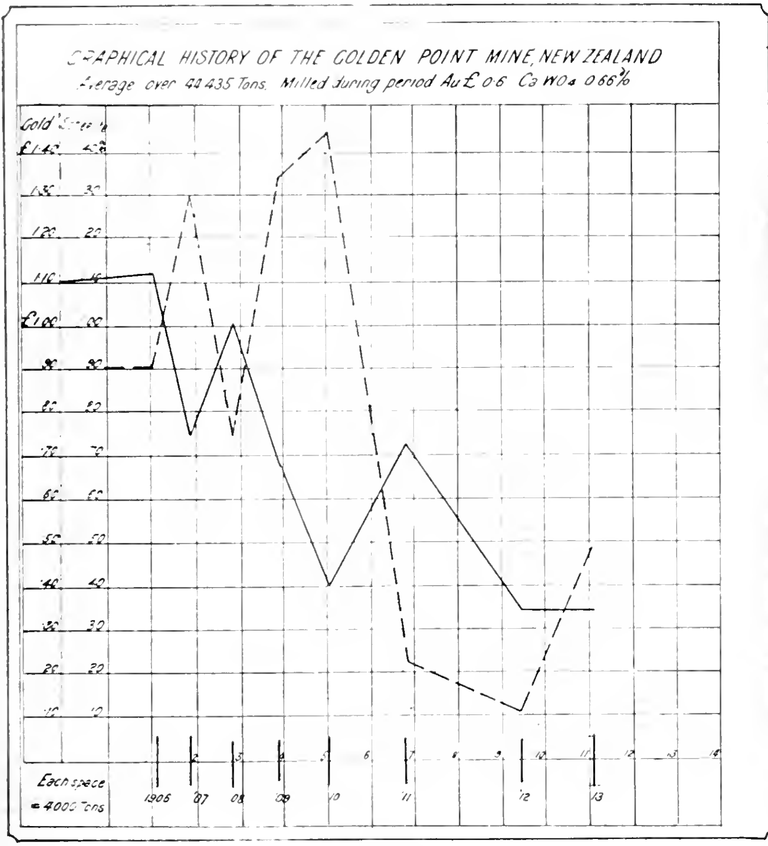


FIG. 6.

The total values produced by this property to the end of 1913 were :—

Scheelite (811 tons)	£68,715
Gold	49,288
Gross total	£118,003

The scheelite obtained from milling operations has been about 30 % of the total scheelite output.

Highlay Mine.

Situated at Macrae's Flat (Fig. 2), 5 miles to the north-west of the Golden Point mine, the Highlay mine started crushing operations in 1893 and continued until March, 1913, when it ceased operations, having exhausted the payable ore-bodies. Two ore-bodies were worked on this property; No. 1 (the lower) was, however, the only one that gave a constant tonnage, and was, in fact, the mine's life. This body strikes N. 30° W., and

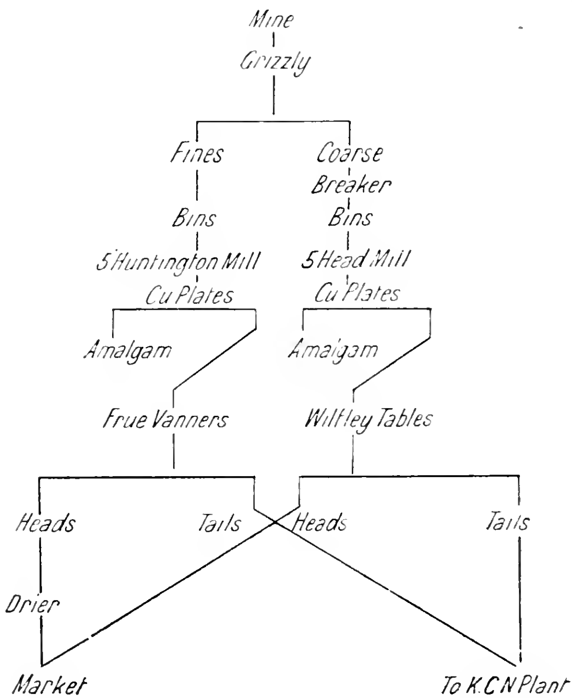


Fig. 7.

Highlay Mining Co. Ltd., Flow Sheet.

underlies N. 30° E. in schist country, was a very much eroded outcrop, and, the erosion having formed a semi-detrital deposit, the ore-shoot in its early stages was worked by open cut; latterly levels were put in to cut the continuation of the body at depth,

with but poor results. This mine for many years had an almost constant production from its 1000 tons per month crushed of 4 tons scheelite (70 % WO_3) and 50 oz. gold—a recovered value of .28 % WO_3, and 3s. 6d. gold per ton. Costs were very low, being somewhere in the region of 3s. 9d. per ton; plant was steam-driven, using coal from its own open-cut coal-pit, costing 1s. 8d. per ton delivered.

In the mill, the management, to get over the excessive sliming of the scheelite, put all fines (which contained quite 60 % of the scheelite values) through a 5-ft. Huntington mill, using the stamps (800 lb.) for the hard and less valuable scheelite ore.

Pulp from the mill passed to a Frue vanner, and from the stamps to a Wilfley table. Direct filling of the KCN vats was tried unsuccessfully, subsequently only the coarser sands being cyanided, slimes being discharged to creek. Flow-sheet (Fig. 7) illustrates the treatment.

These two mines, the Golden Point and the Highlay, represent the extremes in difference of dressing operations in the Macrae's Flat district, all others operating somewhere between the limits defined by the above descriptions.

The Glenorchy Mine.

The property of the Glenorchy Scheelite Mine Ltd. is situated (Fig. 1) on Mount Juda, in the Lake County, adjacent to Lake Wakatipu, 240 miles north-west of Dunedin, at an elevation of 1900–2300 ft. The lode occurs in the same belt of metamorphic-schists as the Macrae's Flat district mines are in, but these schists are much harder and more crystalline, classified as the Kakanui schists of the Great Otago Schist belt; the lode strikes S.E., and dips S. 40° W.; isometric cubes of FeS_2 are very pronounced. The lode, which varies from 1 to 8 ft. in thickness, strikes S. 85° E., and dips at an even inclination of 30°, being a true fissure lode, outcropping for nearly one mile along its strike.

The mine is opened up by drifts from the surface along the pitch of the ore-body, these levels being kept on the ore-body always; levels are run 60–100 ft. apart on the pitch measurement,

rises being put up each 100 ft., and breast stopes brought up from level to level, all waste matter (not ore) in the lode-channel being used as filling. The ground is taken out on stull and head boards, and when stoping operations for each block is completed the remaining open ground is filled in.

Costs are moderately low, and run about 18s. per ton mining and delivery, and 4s. 6d. per ton for milling and dressing operations.

The mill is driven by water power, and has a capacity of 15 to 20 tons per day, but is not, as a rule, worked full time owing to the company acting as a Customs buyer, and treating ore for the score or so of small prospecting concerns in the district.

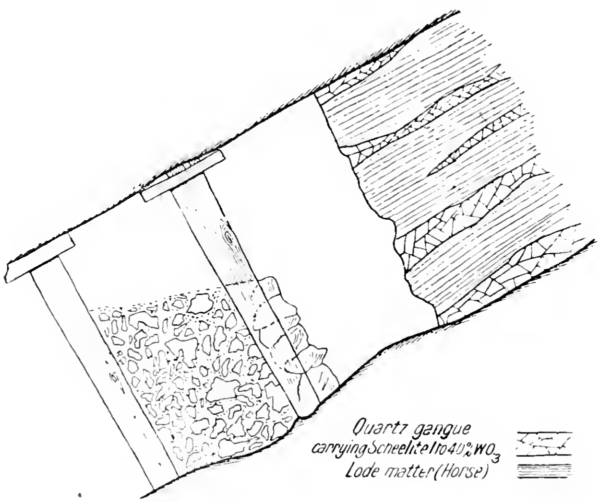


FIG. 8.
THE GLENORCHY SCHEELITE MINING CO. LTD. SECTION BREAST STOPE (1907). SHOWING DEPOSITION OF ORE IN LODE-CHANNEL.
Scale—2 in. to 1 ft.

The gangue is a dense quartz, and has associated with it gold, scheelite, and iron, and arsenical pyrites. The deposition of the scheelite in the ore-channel partakes of the nature of distinct layers (Fig. 8) spaced irregularly through the barren filling or lode-matter. These lenticular masses cut in and out without any

apparent regularity, and vary from .1 to 2 ft. in thickness; all carry high scheelite values, which, within the sulphide zone, become very highly pyritic.

Upon the starting up of milling operations in 1905, the system followed was similar to that obtaining in the Macrae's Flat district — *i.e.*, stamper crushing and concentration, running

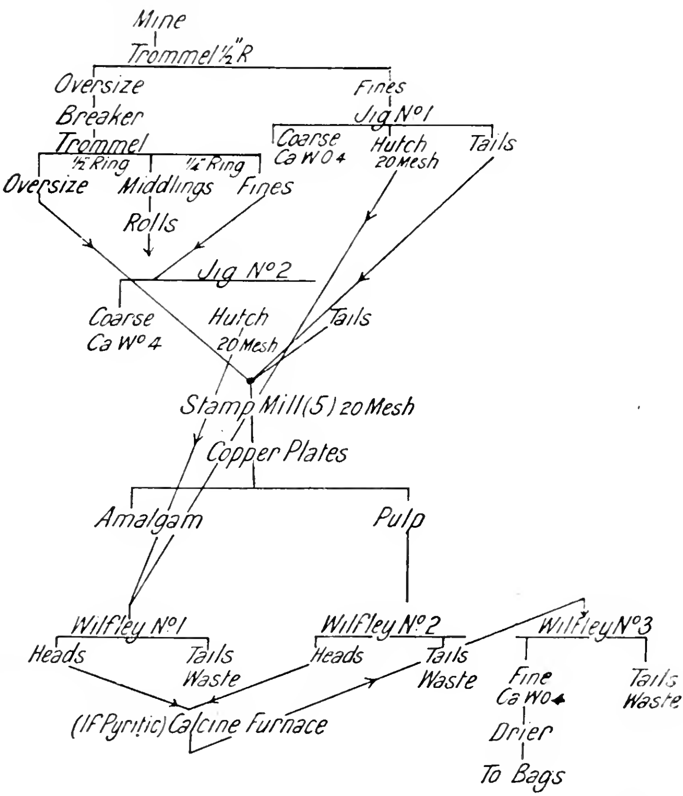


Fig. 9.

Glenorchy Scheelite Mining Co. Ltd., Flow Sheet.

primarily for a scheelite product; but, after going into the loss of slimed scheelite in relation to gold values, the management altered the milling conditions to that shown in flow sheet (Fig. 9), with very excellent results.

The scheelite from this lode being very friable, products are now obtained in the coarse stage from the jigs, the object being to obtain a product while subjecting the ore to the least possible amount of crushing.

The oversize which passes the No. 2 trommel contains, as a rule, very little scheelite, and this, with reject jig tails, is put through the stamp mill, from where it passes to the concentrating floor over amalgamation plates. The hutch (20-mesh) product from the jigs is further concentrated on a Wilfley table. All concentrates, if pyritic, are then roasted in a cylindrical furnace and re-dressed to a 70 % grade WO_3, the iron tails being sent to waste as their gold content is very low.

During 8 years' operations, 1907–14, this mill crushed 5400 tons for a yield of 547 tons (70 % grade) scheelite and 125 oz. gold; this is an extraction value of 7.09 % WO_3 and £.0087 gold per ton crushed, of a total estimated value of £59,138.

Assay for Pyritic Scheelite.

For this class of concentrates the following assay is one that has been adopted at the Golden Point mine, and proved to be highly reliable and accurate:—

Charge, 10 gr. ore. Agate mortar. Digest in aqua-regia and evaporate to dryness three times. Take up with hot H_2O and boil. Filter. Wash with hot H_2O till free of all chlorides. Wash with hot dilute HCl. Dissolve the H_2WO_4 with hot ammonia. Evaporate to dryness. Ignite, and estimate as WO_3.

In conclusion, it may be as well to point out that slightly weathered hand specimens of scheelite and rhodonite are very difficult of determination to even the trained eye, and an engineer will do well, in the examination of any outcropping or weathered lode, to submit his samples to assay determination in any field where rhodonite is present.

Papers and Discussions.

CONTENTS.

NOTES ON FLOTATION OF GOLD AND COPPER ORES, MOUNT MORGAN, QUEENSLAND.

BY W. SHELLSHEAR.

HAVING held for some months, in 1913, the position of metallurgist in charge of the flotation department at Mount Morgan, it occurred to the writer that a description of the research work that he carried out on that ore would be of interest to members of the Institute.

The Mount Morgan ore is virtually a mixture of iron pyrites, copper pyrites, and gold in a quartzose gangue.

There are two types of ore broken in the underground workings —smelting ore containing 40 %–50 % silica, and ore too siliceous for economical smelting.

This latter type of ore was treated by flotation, an approximate analysis being—

Copper	2.0 %	Alumina	1.5 %
Iron	10.0 %	Calcium carbonate, alkalis, &c. ..	3.5 %
Gold	$5\frac{1}{2}$ dwt.	Sulphur	10.0 %
Silica	73.0 %		

[This paper is separately bound, and may be, is so desired, detached complete from this number.]

or, dividing the ore approximately into component minerals—

Copper pyrites	6 %
Iron pyrites	16 %
Quartz calcite, &c. ..	78 %

This analysis is only typical, and does not represent the average assay of the class of ore.

In order to carry on experiments on a large scale, a 6-box standard 24-in. Minerals Separation machine was installed. For three months occasional tests were made with the machine. It was then decided to convert the plant into a large-scale experimental unit treating 100 tons a day, in order that the value of the concentrate produced would, to a large extent, defray experimental costs.

The following is a description of the plant, which is of interest, as it is, to the writer's knowledge, the only dry-crushing plant that has ever been used on a large scale for flotation treatment. The plant was erected on the old West Works chlorination plant site, the same ball mills, coarse crusher, and bins being used as were in use in the West Works before it closed down.

The ore from the shaft bins, approximately 6 in. in size, was hauled in trucks and tipped into a bin of 400 tons capacity. From thence it was fed into a 24-in. x 16-in. Blake-Marsden crusher. The tonnage put through the machine was 10–15 tons per hour, and the horse-power was 20–26.

The crushed product, varying in size from 1½ in. to 2 in., was conveyed along one of the old West Works drier conveyors, and was elevated into bins above the ball mills by bucket elevators.

Eight No. 5 ball mills were put into commission, of which six were constantly in use, travelling at 20 r.p.m.

These were driven by pinion and spur-gearing off one main shaft, and each mill was capable of being thrown out of commission by means of clutch-gearing.

Six-in. steel balls were added regularly with the feed so as to keep the normal weight of 1 ton of balls in each mill.

Screens of 50-mesh were used (not I.M.M. standard screens), the average size of the crushed product being 10 %–15 % + 60 mesh (I.M.M.).

Each mill took 15 h.p. when crushing at its full capacity, which was approximately .9 to 1 ton per hour. The product from the ball mills passed into two fine-feed bins, each capable of holding 150 tons. To prevent escape of fine dust, each bin was lined with sheet iron.

From these latter bins the ore was fed into the M.S. plant by two Challenge feeders delivering into launders, with water pipes

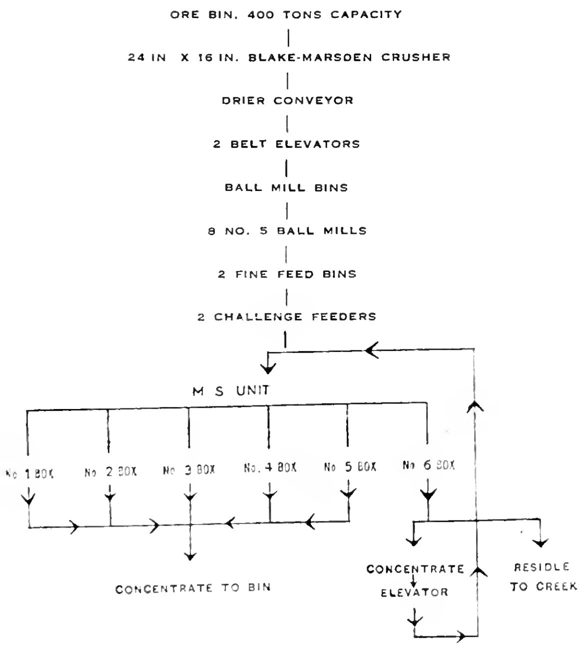


Fig. 1.
Flow Sheet of Plant.

so arranged as to wet the ore with minimum escape of dust. The feeders were very accurate, and could control any required tonnage.

The arrangement for feeding the oil was similar to that at the Kyloe copper plant. The oil was supplied from a tank kept at a constant head by means of a ball valve which actuated a valve in the bottom of a feed tank higher up. The speed of each agitator of the M.S. machine was 295 revolutions, and the whole machine took from 55 to 60 h.p., depending on the rate of feed.

The first 1 to 5 boxes were run for concentrate, and No. 6 box was returned to the feed by means of an elevator.

The concentrate was at first run into round vats, about 20 ft. in diameter and 10 ft. deep, with filter bottoms. These, however, were unsatisfactory owing to the fact that the wet slime always kept to the outside of the vat and did not thoroughly drain. The difficulty was overcome by running the concentrate into a series of rectangular vats, 9 ft. 10 in. by 10ft. 9 in., about 3 ft. deep, with cocoanut matting resting on a sand filter-bed. The concentrate was run through these in series, three or four always being in operation, and each vat was filled right up with concentrate. No trouble whatever was experienced with this system. The drained water was perfectly clear, and the concentrate drained readily in 48 hours to 7 % to 8 % moisture. The residue was run into the creek.

Flotation Experiments.

As a general rule, acid is not used in the flotation of this type of ore, and this particular case was no exception. As typical of one of the many trial tests made on this plant before its conversion into a large-scale unit, the following results were obtained from a special quantitative test on a certain stope of the mine :—

Data of Test.—Rate of feed, 15 tons per hour; dilution of pulp, 3.5 to 1 ; duration of run, 3 hr. ; oil (actual), 1 lb. per ton, eucalyptus being used ; oil was added—$\frac{2}{3}$ to No. 1 mixer, $\frac{1}{3}$ to No. 3 mixer.

Mesh.	Crude Ore.					Concentrate.					Residue.				Recoveries.		
I.M.M.	%	Gold.	Cop'r.	Iron.	Silica.	%	Gold.	Copper.	Iron.	Silica.	%	Gold.	Cop'r.	Iron.	Gold.	Cop'r.	Iron.
		dwt.	%	%	%		dwt.	%	%	%		dwt.	%	%	%	%	%
+60	16.0	7.50	1.24	—	—	6.1	9.79	6.04	36.75	15.6	12.0	5.86	0.67	—	—	—	—
+80	4.9	2.93	1.37	—	—	5.1	8.97	7.12	35.30	17.4	8.0	2.11	0.16	—	—	—	—
+120	18.5	5.86	1.64	—	—	18.1	19.10	8.70	34.47	17.7	12.0	3.09	0.25	—	—	—	—
−120	60.6	9.79	2.65	—	—	70.4	47.50	16.16	32.24	15.15	68.0	1.79	0.21	—	—	—	—
Bulk	100.0	8.64	2.05	9.11	76.25	100.0	33.85	12.08	33.01	17.06	100.0	2.60	0.33	4.27	75.7	86.2	61.0
Actual Recoveries (by weights) ..															69.8	84.2	48.7

Each flotation-box assayed separately gave the following results :—

	Gold.	Copper.	Iron.	Silica.
	dwt.	%	%	%
No. 1 box concentrate ..	55.52	16.28	34.49	10.10
.. 2 ,, ,, ..	44.90	14.36	34.49	12.20
.. 3 ,, ,, ..	40.00	12.84	32.72	17.33
.. 4 ,, ,, ..	41.43	14.38	32.07	16.80
.. 5 ,, ,, ..	25.30	8.10	27.08	32.81
.. 6 ,, ,, ..	18.77	5.31	20.23	50.37

It will be observed that the theoretical recovery of gold in bulk on this test is about 76 %, and, as seen from the sizing tests, it is much higher in the – 120 grade than in any of the coarser grades. The relatively low recovery of gold as compared with that of copper led to a number of tests being carried out to try and find out the exact occurrence of the former.

OCCURRENCE OF GOLD IN VARIOUS FLOTATION PRODUCTS.

Tests on Concentration of Ore by Tabling before Flotation.

Vanning tests, conducted in a dish in the laboratory on various stopes of the mine, indicated that a recovery of 40 % to 50 % of the gold could be obtained, but only about 30 % of the copper, unless the crude ore vanned was very high in copper.

On conducting flotation tests on the vanned residue more gold and copper were recovered.

In the ore a high percentage of copper nearly always indicated a high gold content. The vanning concentrate, however, although much lower in copper than the flotation concentrate, was, as a general rule, higher in gold values.

The following laboratory test conducted on one of the stopes illustrates this :—

Crushing.	Crude Ore.			Vanning Concentrate.				% Extraction.		Flotation Concentrate.				% Extract'n.		Residue.		Total Recoveries.	
Thro' Mesh. I.M.M.	Gold.	Cop.	Silica	%	Gold.	Cop.	Silica.	Gold.	Cop.	%	Gold.	Cop.	Silica	Gold	Cop.	Gold.	Cop.	Gold.	Cop.
	dwt.	%	%		dwt.	%	%				dwt.	%	%			dwt.	%	%	%
40	5.70	2.01	82.6	11.3	19.5	3.66	39.66	38.6	20.6	16.8	10.44	7.8	56.81	30.5	39.5	1.63	0.30	69.1	80.1

NOTE.—Recoveries obtained from weights of concentrates.

These tests indicated strongly that, although the gold in bulk was associated with the copper, on fine crushing it certainly was not so associated.

Tests were then conducted on a Wilfley table, sectional samples being taken every 9 in. along the table to ascertain how the gold followed the various table products.

Feed to table was crushed through 50-mesh screens on ball mills of main plant. Feed was sent direct to table without classification.

Table Details.—Speed, 260; stroke, $\frac{5}{8}$-in.; fall, $\frac{1}{2}$-in. per ft.; rate of feed, $1\frac{1}{4}$ tons per hour (see table, p. 58).

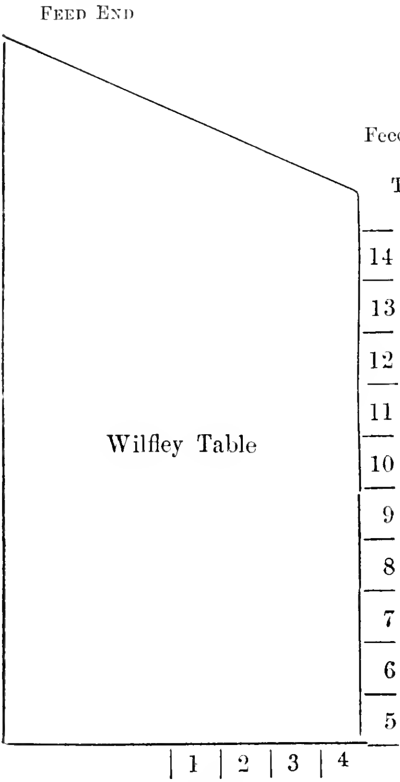
A vanning test was conducted in a dish on a crude sample of sulphide ore assaying 25.14 dwt. of gold. After careful concentration a trace of very fine free gold could be seen on the edge of the dish. Concentrate produced was tested for tellurium, because, on rare occasions, telluride minerals have been found on this mine, but no trace was discovered.

Tests on Occurrence of Gold in Residue after Flotation.

A sample of residue from a large scale flotation test assayed 2.44 dwt. gold, 0.23 % copper. The magnetite was removed carefully under water with a hand magnet, a fairly clean magnetite being obtained. This assayed 3.42 dwt. gold, proving that a certain proportion of the gold is with the gangue.

The proportion of magnetite in the residue only represented a very small percentage of the total iron remaining there.

Sectional Analysis of Wilfley Table treating Crude Ore.



	Gold.	Copper.	Iron.	Silica.
	dwt.	%	%	%
Feed to table	4 72	1·96	8·62	79·39
Table tails	2 69	1 66	6·53	83·98
14	3·74	2·03		
13	1·63	0·82		
12	2·28	0 89		
11	1·30	1·04		
10	1·63	1·14		
9	2·11	1·21		
8	2·60	2·13		
7	2·11	1·58		
6	4·07	3·48	14·10	65·80
5	9·13	4·48	26·11	39·80
4	20·24	4 87	32 56	30·10
3	19·26	5 21	45.78	3·55
2	24·81	3·46	46·10	3·13
1	40·82	2·62	45·78	6·60

Bulk concentrate products, 1 to 5 assayed.

Gold. dwt.	Copper.	Iron. %	Silica.
20.46	4.30	34.1	23.2

Recoveries—Gold, 29·5 %. Copper, 13·3 %.

Test indicates clearly that gold is not associated with copper.

Use of Sonstadt Solution to Determine Whether Gold Occurred in Quartz or Pyrite.

A sample of flotation residue from a large-scale flotation test was taken assaying 4.40 dwt. gold, 0.55 % copper.

Sample was screened through an 80-mesh sieve.

The – 80 product was then screened through 120-mesh.

The sand that remained on the 120-mesh sieve was used for the test. It assayed 5.13 dwt. gold and 0.74 % copper.

Sonstadt solution was prepared by taking a saturated solution of potassium iodide in water and adding mercuric iodide till required specific gravity of solution was obtained. In this case 150 grm. of the sand from the product sieved between – 80 and +120-mesh was added to the Sonstadt solution and stirred well. The Sonstadt solution had to be kept warm to prevent it solidifying.

The quartz product floated on top of the solution, and was skimmed off, filtered free of solution, and dried.

Assays of products obtained were—

	Gold. dwt.	Copper. %	Iron. %	Silica. %
Original product	5.13	0.74	—	—
Quartz product	3.91	0.05	1.45	95.50
Mineral product	9.79	1.85	34.95	18.40

As the mineral product contained 9.79 dwt. of gold, the copper and the iron in the quartz product certainly did not account for the total gold present. Therefore, the test proved that a certain percentage of the gold was associated with the quartz.

In order to obtain still more information on this matter, the original sample of residue was screened through 120-mesh. The – 120-mesh product was then screened through 180-mesh. The sand that remained on the 180-mesh screen was used for the second experiment.

The test was conducted similarly to the first experiment, the quartz product being floated off the top of the Sonstadt solution.

Assays of the quartz product were :—Gold, 1.30 dwt. ; copper, *nil* ; iron, 0.79 %.

This test is of interest, as it clearly indicates that there was less gold associated with the quartz in the finer product. Hence, it seemed reasonable to assume that a certain amount of the gold associated with the quartz was liberated by finer crushing.

Tabling of Residue after Flotation.

Numerous tests were conducted on Wilfley tables in the experimental plant.

The following are results of a typical test :—

Table Details : Sectional Analysis.—Speed, 260 ; stroke, $\frac{5}{8}$-in. ; fall, $\frac{1}{2}$-in. to ft. ; feed, $\frac{3}{4}$ ton per hour. Feed crushed through 50-mesh screens and sent unclassified to table (see table, p. 61).

Average Extra Recovery of Gold by Tabling after Flotation.

Large-scale tabling tests proved that this varied considerably according to the stopes from which the ore was obtained, the recovery in some cases being practically *nil*, in others 10 % on original feed.

The following are the assay values of a typical table concentrate :—

6.85 dwt. gold, 0.66 % copper, 38.3 % iron, 16.8 % silica.

Examination of Flotation Concentrate.

Flotation concentrate was tested for tellurium, but no trace was found. A sample, being the − 80 product of the first flotation-box concentrate, and assaying 47.2 dwt., was vanned to see if any free gold was present. After long and careful vanning very fine free gold could be seen in the dish.

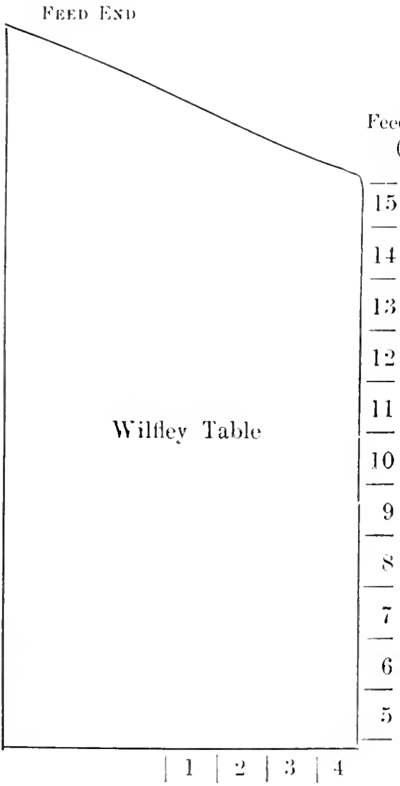
This test proved conclusively that fine free gold floats, and that a certain proportion of it, at any rate, was free in the concentrate.

It seemed reasonable to assume that the gold in the Mount Morgan siliceous ore was free and in a very fine state of subdivision.

FLOTATION OF THE IRON.

As seen in figures of the flotation test, the recovery of iron was low. Results from many runs averaged 45 % to 60 %, the residue assaying 4 % to 8 % of iron. The percentage of iron varied considerably, according to the nature of the ore.

Sectional Analysis of Table treating Flotation Residue.



	Gold.	Copper.	Iron.
	dwt	%	%
Feed to table (calcd.)	4·20	0·53	7·00
15	5·54	0·86	6·21
14	1·95	0·35	4·27
13	1·95	0·30	4·67
12	3·26	0·42	4·27
11	2·60	0·33	4·35
10	3·26	0·42	3·87
9	4·23	0·63	5·24
8	4·72	0·73	7·41
7	5·05	0·87	7·98
6	5·37	0·80	9·03
5	5·05	0·57	7·90
4	4·23	0·45	8·22
3	7·83	0·62	17·81
2	6·69	0·76	18·05
1	14·69	0·60	41·91

The test indicates that gold was in a fine free state distributed through all the table products.

Recovery of Iron in the Various Grades of Crushing.

The following figures were the average results from tests extending over a week at the experimental plant :—

Mesh.	Crude Ore.		Concentrate.	Residue.	Recoveries.
I.M.M.	%	Iron %	Iron.	Iron %	Iron %
+60	15.0	12.09	—	8.27	42.4
+80	4.5	13.03	—	8.58	45.4
+120	15.0	11.46	—	6.63	52.9
−120	65.5	10.53	—	6.24	50.0
Bulk ..	—	11.23	33.85	6.71	50.6

Recoveries of gold and copper during this period were 63.9 % and 80.7 % respectively. The results proved that some of the iron in the crude had no tendency to float even when crushed through 120-mesh.

This was confirmed by numerous laboratory tests on ore crushed through 120-mesh. The average iron in flotation residue was 4 %–5 %.

A sample of flotation residue from large-scale treatment was crushed through 120-mesh and was refloated, giving the following results :—

Product.	Gold. dwt.	Copper. %	Iron. %	Silica. %
Original residue	2.93	0.44	5.77	—
Concentrate	13.71	5.58	21.1	46.14
Residue after re-floating	1.63	0.20	4.98	—
Recoveries on original residue ..	50.4	56.7	18.1	—

The light colour and general appearance of the iron in the residue indicated that it was marcasite which was identi-

fied in this class of ore by Messrs. Newman and Campbell Brown.*

The concentrate from tabling after flotation was found to oxidize very rapidly—usually in about 24 hours—while flotation concentrate did not oxidize, even after standing for days.

This proved that the iron pyrite in the flotation residue was either a different mineral from the iron pyrite in the flotation concentrate or else the oil or some other agent formed a protective coating over the mineral floated. In any case, the proof was sufficient to indicate that most of the iron in the residue had been unaffected by flotation treatment.

EFFECT OF THICKNESS OF PULP.

The results of laboratory tests indicated that, with a dilution of 1 to 1, bad recoveries would be obtained, with thinner dilutions up to 7 to 1, recoveries would be normal.

Rapid Method of Determining Dilution of Feed on Large Scale.

A number of 50 cc. cylinders were filled up to graduated mark with weighed quantities of water and ore, the latter being average crushed feed from experimental plant.

The pulp in the cylinders was then agitated and allowed to settle for five minutes, and the relative number of cubic centimetres of settled feed and of clear water were read off. The results obtained were used as standards for subsequent work.

Tests on the large scale, checking the method against the ordinary one of weighing, showed that it gave good results, the method of taking the dilution being to fill a 100 cc. cylinder with pulp out of one of the agitators. A number of large-scale tests proved that the results were not affected by having dilutions ranging from 2 to 1 and 5 to 1, providing the feed was not heavy.

With a feed of over 20 tons per hour it was found advisable to work with a dilution under 3 to 1, owing to the work being

* See paper No. 204 of Australasian Institute of Mining Engineers.

adversely affected if the rate of flow of pulp through the machine was too great.

Percentage of Slime in Feed to Large Scale Plant.

Dish vanning tests showed that the quantity of elutriated slime in feed varied from 15 % to 20 %.

NATURE OF WATER USED.

The following are average analyses of waters that were used for large-scale experiments. One sample was taken while flowing into the M.S. plant, a second sample from the residue pulp after treatment. Analyses were made on clear solutions after filtering :—

	Before Treatment. Gr. per Gall.	After Treatment. Gr. per Gall.
Total solids	35.14	78.26
Loss on ignition	13.21	23.91
Calcium oxide	4.27	16.87
Magnesium oxide	10.81	9.25
Sulphur trioxide	3.41	27.75
Copper	nil	nil
Iron	nil	nil
Acidity	neutral	neutral

Analyses indicate that $CaSO_4$ is dissolved from the ore during treatment.

The following method of testing whether a particular sample of water was suitable for flotation or not may be of interest :— A sample of crushed ore was treated in the laboratory machine with the water to be tested by passing it dry into the machine and obtaining the recoveries.

A sample of the same ore was then thoroughly wetted with the sample of water and allowed to stand in it for some definite period.

The sample was then treated by flotation, and recoveries so obtained were compared with recoveries from the first test.

The following comparative assays of residues from the same crude ore were obtained from a sample of water which was unsuitable, and which was applied in two different ways. The same conditions of treatment were used in both cases :—

(*a*) Sample treated immediately in water. Residue assayed 2.60 dwt. gold, 0.30 % copper.

(*b*) Sample allowed to remain under water for six hours and then treated by flotation. Residue assayed 3.09 dwt. gold, 0.67 % copper.

Water was found to be neutral, and after agitation with ore still neutral.

EFFECT OF FINE CRUSHING.

The writer regrets that no opportunities were available for finer crushing on the large-scale experimental plant, as tests showed that a screen finer than 50-mesh could not be economically used. Sizing tests on flotation residue produced from ore crushed to 10 % on + 60-mesh (I.M.M.) indicated that the better the crushing the better would be the recoveries.

Experimental tests in the laboratory confirmed this.

As an illustration, the following results were obtained in the laboratory machine on an average sample of ore crushed through 120-mesh.

The crude ore assayed 5.05 dwt. gold, 2.02 % copper. and was treated direct by flotation, 1¾ lb. per ton of eucalyptus being used.

Result :—

Crude Ore.		Flotation Concentrate.				Residue.		Recoveries.	
Gold.	Copper.	Gold.	Copper.	Iron.	Silica.	Gold.	Copper.	Gold.	Copper.
dwt.	%	dwt.	%	%	%	dwt.	%	%	%
5.05	2.02	17.96	7.88	27.62	32.60	1.30	0.09	80.0	96.3

This test showed a marked improvement in recoveries as compared with the best results obtained on this particular class of ore crushed through 50-mesh either in the laboratory or in large-scale plant. The best recoveries with the coarser crushing were 70.0 % gold, 86.0 % copper.

NOTES ON DIFFERENT OILS USED.

Eucalyptus.—Eucalyptus and various blends of eucalyptus with other oils were used in all large-scale tests.

Laboratory tests, backed up by large-scale ones, clearly indicated that, of the eucalyptus oils used, those of the types *Eucalyptus amygdalina* and *Eucalyptus dives*, containing phellandrene, gave better results than those of other types which did not contain phellandrene.

Various tests were made on this point.

Eucalyptus distilled off local plants of the type *Eucalyptus citriodora* and oil distilled from box were tried, but results were not as good as from oils of the phellandrene type just mentioned. In relation to this matter the writer considers it an absolute essential, especially in copper flotation, to test all oils received against a standard oil under standard conditions, and to buy oils on results of testing and results of fractional distillation. Turpentine, which is not able to be distinguished from the first distillate of eucalyptus, is too often used in large quantity as a diluent of eucalyptus, and too much stress cannot be laid on the importance of exact specifications being complied with before eucalyptus in large lots is purchased from suppliers.

Addition of Oil.—In the large-scale plant it was found that the flotation was influenced by an excess of oil almost as much as by too little oil. It was sensitive in this respect, as too much oil caused a heavy float in the residue launder.

In the small-scale laboratory machine, of the ordinary spitzbox type, this was not found to be the case.

Experiments on the large scale, adding oil continuously for a long period on the dry feed in the Challenge feeders, proved that

the percentage of insoluble in the concentrate did not increase above normal.

Experiments were also made to determine the effect of oil in varying quantities to the different agitators. The best results were obtained by adding oil partly in No. 1 and partly in either No. 2 or No. 3 boxes.

Tests with Different Oils.

Many oils were tried in the laboratory, but the writer confines himself to two.

Oleic Acid, in flotation without acid, is interesting in its action, as it floats everything; in fact, so great is its tendency to pick up sand, that a mixture of 5 % oleic acid and 95 % eucalyptus gave a concentrate with over 50 % insoluble.

Residuum Oil.—Although the results obtained by residuum oil alone were only moderate in the laboratory and bad on the large-scale plant, residuum–eucalyptus mixture proved very successful.

In order to compare results obtained in the laboratory with those from actual large-scale experiments, tests are given in each case. In connection with this matter, it is of importance to mention that the eucalyptus which is used must be of a good quality, as a poor brand of eucalyptus cannot be made to give good results by blending it with residuum oil.

A COMPARATIVE LABORATORY TEST OF EUCALYPTUS AND EUCALYPTUS–RESIDUUM MIXTURES.

Samples were crushed through 50-mesh screens in large plant, and treatment conditions were kept as nearly as possible identical in each case.

Laboratory tests, therefore, indicated that an equal weight of mixed oil gave higher recoveries in gold and copper and a reduction in cost of oil, as eucalyptus cost 10d. per lb. and residuum 1¼d. per lb., but the concentrate was higher in insoluble.

Experiment I.

Crude Ore.			Concentrate.			Residue.		Recoveries.		Oil Used.
Gold.	Copper.	Silica.	Gold.	Copper.	Silica.	Gold.	Copper.	Gold.	Copper.	
dwt. 8.81	% 2.27	% 73.15	dwt. 35.26	% 12.05	% 15.80	dwt. 3.42	% 0.53	% 68.5	% 80.0	$1\frac{1}{2}$ lb. per ton eucalyptus.
—	—	—	28.76	9.66	28.70	2.60	0.20	77.6	93.3	$1\frac{1}{2}$ lb. per ton of a mixture of 2 parts eucalyptus and 1 part residuum.

Experiment II.

Crude Ore.			Concentrate.			Residue.		Recoveries.		Oil Used.
Gold.	Copper.	Silica.	Gold.	Copper.	Silica.	Gold.	Copper.	Gold.	Copper.	
dwt. 8.64	% 2.04	% 76.25	dwt. 33.63	% 11.98	% 23.95	dwt. 2.36	% 0.25	% 78.2	% 89.6	$1\frac{3}{4}$ lb. per ton eucalyptus.
—	—	—	31.18	11.34	29.50	1.63	0.25	85.7	89.8	$1\frac{3}{4}$ lb. per ton of a mixture of 2 parts eucalyptus and 1 part residuum.

LARGE-SCALE TESTS COMPARING THE ACTION OF EUCALYPTUS WITH THAT OF EUCALYPTUS-RESIDUUM MIXTURES ON FLOTATION.

Crude Ore.				Concentrate.				Residue.			Recoveries.		Eucalyptus.	Residuum.	Cost of Oil.
Gold.	Copper	Iron.	Silica.	Gold.	Copper.	Iron.	Silica.	Gold.	Copper	Iron.	Gold.	Copper	Lb. per ton.	Lb. per ton.	Per ton in Pence.
dwt.	%	%	%	dwt.	%	%	%	dwt.	%	%	%	%			d.
5.54	2.03	11.25	—	24.49	10.68	30.36	19.52	2.36	0.16	6.67	63.1	80.8	.8		8
5.37	2.01	11.17	—	32.66	15.49	—	15.43	1.95	0.38	7.14	67.6	83.1	.5	.5	5.6
4.89	2.08	11.35	73.48	16.33	9.80	29.13	28.68	1.63	0.21	6.14	71.0	91.9	.1	.8	1.9
5.54	2.00	10.43	73.2	23.83	9.86	33.26	21.67	2.44	0.31	6.54	62.2	85.9	.4	1.4	5.7
4.89	2.12	11.35	72.3	19.91	10.11	30.06	25.13	2.11	0.38	6.51	63.0	85.3	.4	1.2	5.5
5.54	1.98	10.39	74.6	27.76	12.38	33.56	14.70	2.60	0.32	5.97	58.4	86.1	.3	1.2	4.5

The following tests were carried out on the large scale in the experimental plant, the tests averaging about five to six hours each:—

Mixtures of two parts eucalyptus to one part residuum, one part eucalyptus to one part residuum, up to one part eucalyptus to four parts residuum, were successfully used.

For a considerable period eucalyptus and residuum mixtures only were used.

The oil cost during this period was reduced from 9d. to 5d. approximately per ton of crude ore treated.

All tests were carried out under identical conditions, the feed being 15 tons per hour in each case and dilution 3 to 1.

Tests were daily runs in large-scale plant.

The tests proved that eucalyptus–residuum mixtures gave

as good results as eucalyptus, with a saving in the cost of oil. Observations during the runs indicated that oil conditions were not so sensitive with the mixed oils as with eucalyptus. Results were not quite so good on the large scale with eucalyptus–residuum mixtures as in laboratory tests. This was probably due to the time factor, as the time of treatment was less in the large scale plant. This also may account for the fact that the silica in the concentrate was lower in large-scale tests.

The following are assays of each flotation-box using a mixture of one part eucalyptus to three parts residuum :—

	Gold. dwt.	Copper. %	Silica. %
No. 1 box concentrate	39.01	16.08	7.91
,, 2 ,, ,, ...	33.47	14.72	10.56
,, 3 ,, ,, ...	35.92	15.72	11.40
,, 4 ,, ,, ...	32.66	15.64	13.72
,, 5 ,, ,, ...	13.38	6.07	32.60
,, 6 ,, ,, ...	6.19	2.08	69.56

No. 6 box was returned. Sand floated in Nos. 5 and 6 boxes.

Method Adopted in Large-Scale Concentration Plant.

Although in some cases very little extra gold was recovered by vanning and flotation tests as compared with straight flotation tests, in most cases the extra recovery obtained was appreciable, and this fact was really the main one in considering the erection of tables before flotation in the large treatment plant.

Other advantages were :—

(*a*) Recovery of a large proportion of coarse iron concentrates, which, when mixed with finely ground flotation concentrates, would give a product easy to handle.

(*b*) Possibilities of treating smelting ore in the concentration plant.

The method adopted for this plant was, therefore, briefly this :—

Primary crushing to about 40-mesh and tabling.

Re-crushing table rejects to, say, 80 or 120-mesh and treating them by flotation.

Further discussion of this plant is outside the scope of this paper.

Test 1.—Ore crushed through 120-mesh, vanned, and residue from vanning treated by flotation:—

Dish-Vanning Concentrate.					Flotation Concentrate.					Residue.			Recoveries.		Oil.
%	Gold.	Cop'r.	Iron.	Silica.	%	Gold.	Cop'r.	Iron.	Silica.	Gold.	Cop'r.	Iron.	Gold.	Cop'r.	
	dwt.	%	%	%		dwt.	%	%	%	dwt.	%	%	%	%	
18.7	15.01	3.13	40.25	12.84	29.0	16.33	9.73	27.16	33.96	2.28	0.52	6.36	88.4	93.2	Eucalyptus, $1\frac{3}{4}$ lbs. per ton.

SMELTING ORE.

Tests showed that this also required very fine crushing to get good recoveries. As no work was done with the large-scale experimental plant on this class of ore, the writer confines himself to briefly giving some laboratory tests, comparative tests on dish vanning and flotation, and straight flotation on smelting ore.

Crude ore sample assayed 9.37 dwt. gold, 3.42 % copper, 19.0 % iron. 55.34 % silica.

Test 2.—Ore crushed through 120-mesh and treated by straight flotation:—

Flotation Concentrate.				Residue.			Recoveries.		Oil.
Gold.	Copper.	Iron.	Silica.	Gold.	Copper.	Iron.	Gold.	Copper.	
dwt.	%	%	%	dwt.	%	%	%	%	
16.33	7.20	36.47	16.08	3.58	0.67	8.22	79.3	85.3	Eucalpytus 2 lb. per ton.

Results for weekly period ending 6/9/13.

Mesh.	Crude Ore.					Concentrate.					Residue.				Recoveries.	
I.M.M.	%	Gold.	Cop'r.	Iron.	Silica.	%	Gold.	Copper.	Iron.	Silica.	%	Gold.	Cop'r.	Iron.	Gold.	Cop'r.
		dwt.	%	%	%		dwt.	%	%	%		dwt.	%	%	dwt.	%
+60	16.7	5.54	1.10	—	—	11.0	21.38	7.30	—	—	23.3	4.07	0.75	—	—	—
+80	4.7	5.21	1.32	—	—	6.6	19.28	8.79	—	—	6.6	3.26	0.47	—	—	—
+120	27.0	4.23	1.75	—	—	40.2	26.12	11.48	—	—	23.3	1.95	0.22	—	—	—
−120	51.6	7.67	2.44	—	—	42.2	40.82	13.23	—	—	46.8	1.63	0.19	—	—	—
Bulk	100.0	6.03	1.90	10.81	74.30	100.0	31.02	11.75	32.21	19.40	100.0	2.44	0.35	6.78	64.6	84.0

The tests show the advantage of combined treatment.

The high percentage of iron in the residue should be noted.

The brief summary of results (p. 72) which were obtained by the experimental plant when worked on the large scale may be of interest.

Results for monthly period ending 19th October, 1913, were :—

Crude ore milled—tons daily, 83.

Breaker—average rate, 12 tons per hour.

Ball mills, average rate, .98 tons per hour per mill.

Flotation plant, average rate, 17.3 tons of feed per hour.

	Gold. dwt.	Copper. %
Crude assayed	6.01 ..	2.10
Concentrate	26.16 ..	10.66
Residue	2.50 ..	0.48
Actual recoveries	68.58% ..	80.00

The following figures show the actual costs of complete treatment of the ore during the period including trucking from the shaft-bins to the mill and delivery of the concentrates to the smelting works :—

	s.	d.
Haulage		8.94
Breaking		6.86
Milling	2	2.88
Flotation	1	5.18
Residue		.13
Handling concentrate		3.83
Sundries		.27
	5	4.09 per ton

Flotation costs were divided up as follows :—

		d.
Supervision		.9
Wages operating		1.82
Power		1.46
Stores, general		.10
Oil (eucalyptus and residuum) ..		4.98
Maintenance		2.85
Plant and general maintenance		.62
Sampling and assaying		4.45
	1	5.18

In conclusion, the writer desires to thank Mr. A. A. Boyd, general manager of the Mount Morgan Gold Mining Company Ltd., for his permission to read this paper before the Institute.

ANALYSIS OF MINERAL WATERS BY VOLUMETRIC METHODS.

BY E. BROUGHTON JENSEN.

THE mine analyst, or assayer, as he is usually called, is frequently asked to determine the composition of mineral waters for battery, boiler, leaching, or other purposes. The tabulation of his results is generally a matter of taste, and is arranged in accordance with the wishes of the management; but it is usual to supplement the list of bases or metals and acid radicals with a statement showing the computed probable composition of the various salts in solution.

Although it is some ten years since the writer was employed as a mine assayer, the many advantages of volumetric work as compared with gravimetric methods recently tempted him to investigate the estimations of the metal Mg and the radical SO_4 volumetrically in connection with the examination of boiler waters. In connection with these researches an extensive series of experiments indicated conclusively that the volumetric methods herein described, *inter alia* (two methods for Mg and one for SO_4), give results with certainty and regularity to within 0.2 % MgO and SO_4 respectively.

It is proposed to describe herein the methods adopted by the writer for the complete determination of all salts usually met with in mineral waters which come under the notice of the mining engineer.

The mine assayer is seldom called on to examine water from an hygienic point of view, and, as far as the inorganic analysis of a mineral water is concerned, the following eight operations will usually cover the ground:—

(*a*) TOTAL SOLIDS.
(*b*) SILICATES.
(*c*) ALKALIES.
(*d*) CARBONATES.
(*e*) CHLORIDES.
(*f*) CALCIUM.
(*g*) SULPHATES.
(*h*) MAGNESIUM.

[This paper is separately bound, and may be, if so desired, detached complete from this number.]

It is not recommended that the estimations be carried out in the sequence in which they are here described. The chlorides, carbonates, and sulphates are assays requiring but little time, and are best performed during the evaporation of the total solids, as the information thus obtained will be of assistance in operation (*c*) and (*g*). The calcium determination is also a short one, and should be followed by that for magnesium, which can be completed by the time the operator is half way through the alkalies.

(*a*) Total Solids.

It is common practice to determine total solids in a comparatively small portion of the sample (as much as a platinum dish will hold), and to obtain the silicates and alkalies from the filtrate from magnesium assay. But the small quantity of silicates usually present in the water, the desirableness of handling a fair bulk when dealing with the alkalies, and also the advantage of greater accuracy in the determination of the total solids themselves, leads one to recommend operating on not less than one litre of the sample in all cases where waters contain less than 100 gr. of dissolved solid matter per gallon. The advantage of working on a considerable quantity when examining the alkalies results from the fact that it is difficult to obtain reagents free from these compounds. The slight increments which result during separation are liable to amount to a considerable percentage at the finish unless the total weight of the alkalies present in the assay is reasonably great. The greater bulk of solid matter obtained upon evaporation also offers the advantage that the small increases in weight, changes in colour, tendency to partially fuse or spit, &c., are more readily ascertained, and serve as a useful guide to the operator, assisting him in the final interpretation of his results.

Modus Operandi.—For the assay take one litre of the water and evaporate down to dryness. The method of conducting the evaporation will depend upon the apparatus available, but the final operation should be carried out where possible in a platinum dish, finishing to dryness in a water bath at 100° C. Cool under

a desiccator, weigh, and record the result. Allow to remain exposed to the atmosphere for a while and again weigh; if an increase of weight is observed it points to the presence of chloride of calcium or magnesium, or both, and the fact should be noted. Then heat to 200° C. in an air bath, cool under a desiccator, and again weigh, noting the weight, which will be approximately the anhydrous weight of the salts in the sample, with which the assayer will endeavour to balance his results. Then heat to bright redness and carefully examine the residue. Partial fusion will indicate the presence of alkaline carbonates, which are, however, rare, and are not found in waters containing sulphate of magnesium. The weight should again be recorded; the loss, except when nitrates occur, is a rough measure of the CO_2 present in combination with calcium and magnesium, and is useful in so far as it denotes whether much or little of the total Ca and Mg (to be determined later) is present in the form of carbonate.

(*b*) Silicates.

Silicates are generally found only to a limited extent in waters rich in other minerals, as, in a more concentrated form, they act as precipitants, tending to soften the water. The determination either as SiO_2 or SiO_3 is best made upon the total solids obtained as described on p. 76. The chemistry of the ordinary method is shown by the following equations:—

$$Na_2SiO_3 + 2HCl = 2NaCl + H_2SiO_3.$$
$$H_2SiO_3 = H_2O + SiO_2.$$

Modus Operandi.—Without removing the total solids from the platinum dish in which they were collected by operation (*a*), take them up with 5 cc. HCl in 20 cc. of distilled water, and evaporate the solution to dryness, finishing over steam at 100° C. to avoid spitting. Take up again with 5 cc. of strong HCl and re-evaporate to dryness, first at 100° C., and then ignite, cool and dissolve out the alkalies and alkaline earths in hot water acidified with a few drops of HCl. Filter off the residue, consisting of SiO_2, wash thoroughly with hot water, and save the filtrate for the alkali determinations. Dry the filter, ignite and weigh the

SiO_2. The weight of SiO_2 in grammes obtained from a litre assay as described, multiplied by a factor of 88.73, will give the result in grains of SiO_3 per gallon. If the results are desired as SiO_2 or Na_2SiO_3, factors are readily calculable.

(c) Alkalies.

Under the heading of alkalies determinations of sodium and potassium are made, either as Na_2O and K_2O or as Na and K, or, in those cases where the chlorides only are present, direct as NaCl and KCl. The assay may be conducted on the filtrate from the magnesium estimation, but preferably upon the filtrate obtained from operation (*b*). The satisfactory performance of the alkali estimations demands considerable care and the use of pure reagents, as salts of the alkalies present in the reagents used during the process of separation would vitiate the final result. If pure chemicals are not readily obtainable the alkali equivalents of the various precipitants used must be determined, and known quantities of the reagents added, so that corrections may be applied when titrating with $AgNO_3$ in the final operation. The process consists of the titration of the mixed chlorides of sodium and potassium with $AgNO_3$, and from a knowledge of the weight of the mixed chlorides, calculating the weight of each present in the assay. It is necessary to remove all other acid radicals except Cl, and all other bases except Na_2O and K_2O.

The chemistry of the operations involved will be apparent from the following equations:—

$$Na_2SO_4 + BaCl_2 = 2NaCl + BaSO_4.$$
$$MgCl_2 + Ca(HO)_2 = CaCl_2 + Mg(HO)_2.$$
$$CaCl_2 + (NH_4)_2CO_3 = CaCO_3 + 2NH_4Cl.$$
$$AgNO_3 + NaCl = AgCl + NaNO_3.$$

Modus Operandi.—Take the filtrate from the assay for silicates, previously described, and remove SO_4 by adding sufficient $\frac{N}{2}$ $BaCl_2$ solution to completely precipitate all sulphates as $BaSO_4$; the number of cubic centimetres of semi-normal $BaCl_2$ solution required are obtainable from the determination of sulphates

described under operation (*g*). The bulk of the assay at this stage should not exceed about 50 cc., and too great an excess of the barium solution should be avoided. Before adding the $BaCl_2$ the assay should be brought to the boil, and the boiling continued for a few minutes after the precipitation has been made. Filter and wash thoroughly, dry, ignite, and weigh the $BaSO_4$ as a check on the sulphate method given on p. 83. The filtrate will now probably only contain chlorides of the alkalies and alkaline earths, together with unimportant traces of metallic chlorides. Remove magnesium and traces of iron present by boiling with pure milk of lime. Filter and wash thoroughly, and then remove calcium from the filtrate by making alkaline with NH_4HO, boiling and precipitating with strong $(NH_4)_2CO_3$ solution, filter and test a small portion of the filtrate with a drop or two of $(NH_4)_2C_2O_4$ to confirm the removal of the calcium. The ammonium compounds are now removed by carefully evaporating to dryness at 100° C. and then igniting at a dull red heat until fumes cease to come off and the weight remains constant. Book the weight of the residue, which should consist of a mixture of NaCl and KCl. Re-dissolve this in 50 cc. of distilled water, and titrate against $\frac{N}{10}$ $AgNO_3$, 1 cc. of which = 0.00355 grm. Cl. This gives the total weight of Cl present in the mixture.

As the molecular weights of NaCl and KCl are known, and also the weight of the mixture and the weight of the Cl in that mixture, it is a matter of simple algebra to calculate separately the amounts of KCl and NaCl, thus:—

$$\text{NaCl contains } \frac{35.5}{58.5} \text{ of its weight of Cl.}$$

$$\text{KCl contains } \frac{35.5}{74.5} \text{ of its weight of Cl.}$$

Let the weight of the mixture be "M" grm

Let the weight of Cl be "C" grm.

Let the required weight of NaCl be "x" grm.

Let the required weight of KCl be "y" grm.

Then $x + y = M$ and $\frac{35.5}{58.5}x + \frac{35.5}{74.5}y = C$.

or

$$.6x + .47y = C \text{ and } x = \frac{C - .47y}{.6}$$

Substitute the value of x in equation $x + y = M$, then $\frac{C - .47y}{.6}$ $+ y = M$. Mutiplying across by .6 and extracting y,

$$\text{Weight of KCl} = y = \frac{.6M - C}{.13}$$

$$\text{Weight of NaCl} = x = \frac{C - .47M}{.13}$$

This gives the result in grammes per litre. To convert to grains per gallon multiply by 70.05.

From the weights of NaCl and KCl thus found, the results can be easily calculated to Na_2O and K_2O or Na and K, as may be desired.

(*d*) Carbonates.

The estimation of the carbonates incidentally determines the temporary hardness of the water, except in those unusual instances in which carbonates of the alkalies are present, in which case the temporary hardness is determined by the number of grains of magnesium and calcium carbonates as computed in the final interpretation of the assayer's results. If the water is found to contain sulphates or chlorides of magnesium or calcium, it is fair to assume that no carbonates of the alkalies are present, and the CO_2 found in this assay expressed in terms of grains of $CaCO_3$ per gallon gives the measure of the temporary hardness of the sample. The carbonates are investigated by direct titration with $\frac{N}{10}$ HCl, using methyl orange as an indicator. The assay is best performed in daylight, as the finishing point is somewhat masked by artificial light. Although the calcium carbonate is present

in waters actually in the form of $H_2Ca(CO_3)_2$, and the magnesium salt in a corresponding form, the results are always reported as $CaCO_3$ and $MgCO_3$ in grains per gallon.

Modus Operandi.—For the assay take 350 cc. of the sample, add one drop of methyl orange (merely sufficient to impart a slight yellow tinge to the water). The end reaction is less marked in the presence of too much of the indicator. Titrate cold against deci-normal HCl. If the end point is a little unsatisfactory, it is well to overdo the finish and bring the assay back with $\frac{N}{10}$ Na_2CO_3, and to repeat this procedure until the operator is satisfied with the finish. The number of cubic centimetres of acid required will then be the difference of the readings of the acid and alkali burettes.

To reduce the result to CO_3 in grains per gallon, multiply the number of cubic centimetres of $\frac{N}{10}$ HCl required by 0.6, or the result may be reported as CO_2 in grains per gallon by multiplying by 0.44, or as $CaCO_3$ grains per gallon by multiplying by 1.0.

(*e*) Chlorides.

By titrating in neutral solution with deci-normal $AgNO_3$ the method is rapid and accurate; but the assay must be neutral, and if acid (as sometimes occurs in mine waters), the sample should be first boiled with $CaCO_3$ (freshly precipitated) and then filtered. Standard $\frac{N}{10}$ $AgNO_3$ is prepared by dissolving 17.0 grm. of pure, dry silver-nitrate in water and making up to one litre. Two or three drops of a concentrated solution of K_2CrO_4 are added before titrating to indicate the end point by the formation of red chromate of silver.

Standard $\frac{N}{10}$ NaCl, required for checking the $AgNO_3$ and as a check on the finish, is prepared by precipitating from a concentrated solution of brine by means of strong HCl, filtering,

washing with brine, drying first on a porous plate, and then at 100° C., and dissolving 5.85 grm. in a litre of water.

The principle of the $AgNO_3$ titration is as follows:—

$$AgNO_3 + KCl = KNO_3 + AgCl.$$

Modus Operandi.—For the assay take 70 cc. of the sample and bring to exact neutrality. To do this it may be necessary to determine the exact amounts of acid or alkali required for the purpose by conducting a separate experiment in the presence of suitable indicators. To the neutral solution add two or three drops of concentrated K_2CrO_4 solution, which should just suffice to impart a greenish-yellow colour to the assay, and titrate cold with the $AgNO_3$ $(\frac{N}{10})$, adding the silver standard cautiously until the pink tinge which marks the finish persists. The first sign of change of colour marks the end point, and the reading should be noted. As a check it is then well to overdo the titration and bring back with $\frac{N}{10}$ NaCl solution. The number of cubic centimetres of $AgNO_3$ required multiplied by 3.55 will give the result in grains of Cl per gallon.

(*f*) Calcium.

The calcium is precipitated from a hot ammoniacal solution by means of $(NH_4)_2C_2O_4$: the precipitate is dissolved in H_2SO_4 and titrated with $KMnO_4$ of deci-normal strength.

Standard $\frac{N}{10}$ $KMnO_4$ is prepared by dissolving 3.15 grm. of the dry crystals and making up to one litre, and is standardized against pure iron-wire, 0.2 grm. of which should require exactly 35.7 cc. of the permanganate. Or the standardizing may be carried out with $FeSO_4$, $(NH_4)_2SO_4$, $6H_2O$ crystals, which compound contains just one-seventh of its weight of Fe.

$$\begin{aligned} 1 \text{ cc. of } KMnO_4 &= 0.0028 \text{ grm. } CaO. \\ &= 0.005 \text{ grm. } CaCO_3. \\ &= 0.0056 \text{ grm. Fe.} \\ &= 0.002 \text{ grm. Ca.} \end{aligned}$$

Modus Operandi.—Take 350 cc. of the sample, acidify with a few drops of HNO_3, and boil down to about 250 cc. Add 1 grm. of NH_4Cl, and make alkaline with NH_4HO. Continue the boiling while 10 cc. of a concentrated solution of $(NH_4)_2C_2O_4$ are stirred in. Allow to settle, and filter while hot, washing with hot distilled water three or four times to remove oxalates. The percipitate is then dissolved in hot dilute H_2SO_4 and titrated at 70° C. with the standard $KMnO_4$. When getting the precipitate into solution it is best not to prick a hole in the filter, but to flatten out the paper against the funnel and wash off, as the presence of even a small amount of fibre is sufficient to affect the permanganate titration.

In order to convert the result into grains of Ca per gallon, multiply the number of cubic centimetres of $\frac{N}{10}$ $KMnO_4$ required by 0.4. To obtain the result as CaO, multiply by 0.56, and to report as $CaCO_3$ multiply by 1.0.

(*g*) Sulphates.

The volumetric method here given is reliable and consistent, except in the presence of nitrates, which interfere; these compounds, however, rarely occur in waters met with by the mining engineer. The chemistry of the method herein described is not new, but the process has been adapted, from a brief description given in Sutton's book, to suit the conditions of water analysis, and, as it has yielded the writer excellent results when working against known quantities, it is recommended with confidence.

A known volume of $\frac{N}{2}$ $BaCl_2$ solution is made use of to precipitate the SO_4 in the sample:

$$BaCl_2 + CaSO_4 = CaCl_2 + BaSO_4,$$

and the exact amount of $BaCl_2$ required for that purpose is ascertained by titrating the excess with $\frac{N}{2}$ K_2CrO_4—

$$BaCl_2 + K_2CrO_4 = BaCrO_4 + 2KCl.$$

The end reaction in this titration is readily determined by the slight yellow tinge which an excess of one drop of K_2CrO_4 imparts to the assay.

Standard $\frac{N}{2}$ $BaCl_2$ solution is prepared by dissolving 61 grm. of the pure crystals of $BaCl_2$, $2H_2O$ in water and making up to a litre.

$$1 \text{ cc. } \frac{N}{2} \text{ } BaCl_2 = 0.052 \text{ grm. } BaCl_2.$$
$$= 0.024 \text{ grm. } SO_4.$$

Standard $\frac{N}{2}$ K_2CrO_4 is obtained by dissolving 48.5 grm. of the crystals in water and making up to a litre. These solutions should be tested against each other; 1 cc. of $BaCl_2$ solution should neutralize 1 cc. of the K_2CrO_4 solution as made up.

Modus Operandi.—Take 350 cc. of the sample, add sufficient $\frac{N}{10}$ HCl to bring to exact neutrality (the amount required will have been determined in operation "*d*"); then boil down to about 200 cc., and while boiling add 10 cc. of $\frac{N}{2}$ $BaCl_2$ solution (unless more $BaCl_2$ is required to completely precipitate the SO_4). For the sake of accuracy the $BaCl_2$ should be run in from a burette. Continue the boiling for a couple of minutes and titrate at 70° C. with $\frac{N}{2}$ K_2CrO_4 solution until the end reaction is shown by the yellow tinge of the supernatent liquid after allowing the precipitate to settle. If the settling is slow the assay may be hastened by filtering off a few drops of the liquid into a test tube and noting the colour of the filtrate, returning the filtrate to the bulk of the assay when this has been done. For rapidity it is preferable to run in the chromate solution moderately quickly, overdoing the titration by half a cubic centimetre or so, and then to bring it carefully back drop by drop with $\frac{N}{2}$ $BaCl_2$ solution until the colour, as seen in a small filtered portion, is exactly

discharged. The quantity of $BaCl_2$ actually required for bringing down the SO_4 of the assay will be the difference between the total amount of $BaCl_2$ solution taken and the reading of the chromate burette.

To obtain the result in terms of SO_4 grains per gallon, multiply the number of cubic centimetres of $BaCl_2$ used in the precipitation by 4.81. Or the result may be reported as SO_3 or as S by calculating the necessary factor.

(*h*) Magnesium.

By whatever method it is decided to carry out the magnesium determination, the filtrate from the calcium assay should be used, *vide* operation (*f*). The writer investigated a volumetric method based on Stolba's process, except that Na_2HAsO_4 was used as the precipitant instead of Na_2HPO_4, and also an iodimetric method, which, although founded on the well-known arsenic assay, he believed to be new in its application to magnesium determinations. Both methods, which will be described herein, have yielded good results, but the iodine process is probably slightly less liable to error than the other.

I.—ALKALIMETRIC METHOD FOR MAGNESIUM.

This method presents no novelty, except that the writer recommends the precipitation of the magnesium in the form of $MgNH_4AsO_4$ instead of the corresponding phosphate, on the grounds that the arsenic compound offers advantages. Na_2AsHO_4 is added to the Mg solution in the presence of NH_4HO and NH_4Cl, and the resulting double arsenate is attacked with $\frac{N}{10}$ HCl, and the amount of HCl absorbed is found by titrating the excess with $\frac{N}{10}$ Na_2CO_3. The chemistry of the process will be apparent from the following:—

$$Mg(HO)_2 + 4NH_4Cl = 2NH_4HO + MgCl_2.2NH_4Cl.$$
$$MgSO_4 + Na_2HAsO_4 + NH_4HO = MgNH_4AsO_4 + Na_2SO_4 + H_2O.$$
$$MgNH_4AsO_4 + 2HCl = MgCl_2 + NH_4H_2AsO_4.$$
$$Na_2CO_3 + 2HCl = 2NaCl + H_2O + CO_2.$$

Standard $\frac{N}{10}$ Na_2CO_3 is prepared by dissolving 5.3 grm. of pure anhydrous Na_2CO_3 in water and making up to 1 litre. Standard $\frac{N}{10}$ HCl is prepared by titrating a known volume of, say, 1 to 1 HCl with the standard alkali, and ascertaining the exact amount of water required to be added in order to reduce its strength so that 10 cc. of $\frac{N}{10}$ Na_2CO_3 shall exactly neutralize 10 cc. of the HCl solution.

$$1 \text{ cc. of } \frac{N}{10} \text{ HCl} = 0.0012 \text{ grm. Mg.}$$
$$= 0.006 \text{ grm. } MgSO_4.$$
$$= 0.002 \text{ grm. MgO.}$$

The Na_2HAsO_4 solution should preferably be of known value, a convenient strength being 1 cc. = 0.005 grm. MgO. This solution is prepared by dissolving 12 grm. of As_4O_6 in HNO_3 and evaporating to dryness at 100° C., dissolving the residue in 200 cc. of warm Na_2CO_3 solution and then making up to 1 litre with distilled water.

Modus Operandi.—Transfer the filtrate from the calcium assay to a stoppered bottle, cool, add 70 cc. of NH_4HO and allow to stand for a minute. If a precipitate forms, add sufficient NH_4Cl to dissolve it; if no precipitate appears, sufficient NH_4Cl will have been carried over from the calcium assay to prevent the formation of $Mg(HO)_2$. Then add 70 or 80 cc. of a_2HAsON_4 solution, of the strength described above. Allow to stand for a minute or two, and then shake vigorously for 10 minutes, when the precipitation of the magnesium will be complete. The precipitate should be crystalline, and settle readily, its composition at this stage being $MgNH_4AsO_4,6H_2O$. Filter and discard the filtrate. Wash out the bottle in which the precipitation was made with 50 cc. of 1 to 3 NH_4HO and water, and put the washings through the filter, adding small portions at a time. Then make up 60 cc. of 1 to 1 alcohol and water to dissolve out the ammonia, using at a time say 15 cc. for rinsing out the bottle and washing

the residue with same, taking care to allow the washings to drain completely through the filter before adding more. When free from ammonia dissolve the precipitate in a carefully-measured volume of 50 cc. of $\frac{N}{10}$ HCl nearly boiling, getting the solution into the bottle in which the precipitation was made. Shake up and allow to cool, and titrate cold in the presence of methyl orange, with $\frac{N}{10}$ Na_2CO_3 to determine the uncombined excess of $\frac{N}{10}$ HCl.

The number of cubic centimetres of Na_2CO_3 required, deducted from the 50 cc. of HCl taken, will give the measure of HCl absorbed.

The number of cubic centimetres of HCl absorbed, multiplied by 0.244, will express the result in grains of Mg per gallon; or, multiplied by 0.407, as grains of MgO per gallon.

2.—IODIMETRIC METHOD FOR MAGNESIUM.

In this process the double ammonium magnesium arsenate is obtained as described above and reduced to arsenite by digestion with SO_2 solution, and the assay then titrated with $\frac{N}{10}$ iodine solution.

The oxidizing effect of the iodine is illustrated thus:—

$$MgHAsO_3 + I_2 + H_2O = MgHAsO_4 + 2HI,$$

from which

$$\begin{aligned} 1 \text{ cc. of } \tfrac{N}{10} \text{ I solution} &= 0.002 \text{ grm. MgO.} \\ &= 0.0012 \text{ grm. Mg.} \\ &= 0.0061 \text{ grm. } MgSO_4. \end{aligned}$$

Standard $\frac{N}{10}$ iodine solution is prepared by carefully and rapidly weighing out 12.7 grm. of iodine in a weighing bottle and dissolving in about 200 cc. of distilled water, to which 18 or 19 grm. of KI have been added, then making up to 1 litre. As a check, this

solution may be titrated with either $\frac{N}{10}$ $Na_2S_2O_3$ (24.8 grm. of crystals per litre), or preferably with As_4O_6—

$$As_4O_6 + 4I_2 + 4H_2O = 8HI + 2As_2O_5.$$

For this purpose dissolve 4.95 grm. of As_4O_6 in about 400 cc. of distilled water with 30 grm. of $NaHCO_3$, warmed up, but not to such a temperature as will decompose the bicarbonate into the normal salt; cool, and make up to 1 litre. When titrating with iodine an excess of alkali in the form of bicarbonate is required to neutralize the HI formed in the process.

The SO_2 solution is prepared by adding to a known weight of copper chips in a flask three times its weight of concentrated H_2SO_4 and heating.

$$Cu + 2H_2SO_4 = CuSO_4 + SO_2 + 2H_2O.$$

Pass the evolved SO_2 into a bottle with a narrow neck, as illustrated in Fig. 2. At 70° Fahr. one volume of water will dissolve 40 volumes of SO_2; complete saturation, however, is not necessary for the purpose in hand. A sufficient strength is such that 5 cc. of the SO_2 water will require about 50 cc. of $\frac{N}{10}$ iodine solution for its complete oxidation. With a pipette draw off 5 cc. of the SO_2 water in course of preparation, neutralize with NaHO with phenol as indicator, slightly acidify with $H_2(C_4H_4O_6)$, and titrate with $\frac{N}{10}$ iodine. Or the strength of the SO_2 solution may be determined by titrating with $\frac{N}{10}$ $KMnO_4$ solution.

$$2KMnO_4 + 5H_2SO_3 = K_2SO_4 + 2MnSO_4 + 2H_2SO_4 + 3H_2O.$$

Modus Operandi.—Work on the filtrate from the calcium assay and proceed exactly as described in the first portion of the alkalimetric method up to the point at which the precipitated double arsenate is freed from ammonium. The precipitate is then dissolved in 100 cc. of SO_2 water and transferred to the stoppered bottle in which the precipitation was carried out. Fasten down the stopper with a piece of string and digest for half an hour in a bath of boiling water; the actual apparatus used by the writer

is shown in Fig. 1. If time is no object the tying down of the stopper and subsequent digestion may be dispensed with, if the bottle with the stopper in is allowed to stand for two hours, which will then suffice to complete the reduction from arsenate to arsenite. After reduction transfer the liquid to a 500-cc. beaker, add 200 cc. of distilled water, and boil vigorously down to a bulk

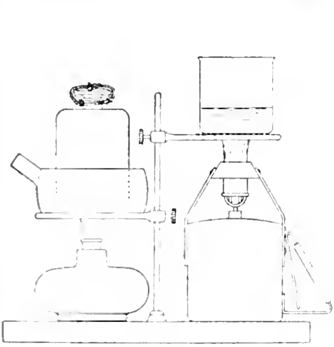


FIG. 1.

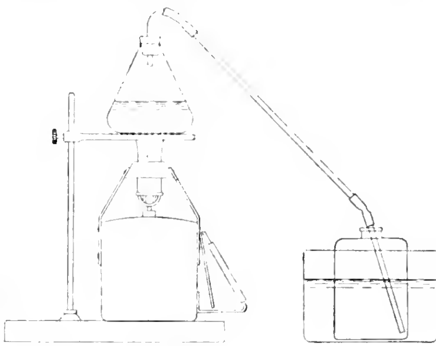


FIG. 2.

of about 100 cc. It is important that the boiling be vigorous, and not a gentle evaporation, to ensure the removal of all surplus SO_2. Cool, and neutralize with NaHO, using phenolphtalein as the indicator, then make slightly acid with $H_2C_4H_4O_6$ and add 5 grm. of solid $NaHCO_3$ or $KHCO_3$ (avoid the normal carbonates, as they affect the so-called blue iodide of starch). Add a few drops of starch solution and titrate cold with the standard $\frac{N}{10}$ iodine solution. The finish is obtainable to half a drop. The result may be reported as grains of Mg per gallon by multiplying the number of cubic centimetres of standard iodine required by 0.244, or as grains of MgO by multiplying by 0.407.

(*i*) HARDNESS OF WATER.

A knowledge of the temporary and total hardness of the sample is of assistance to the operator in connection with the final computation of results. In fact, any test or process which will be

helpful in that matter should be undertaken, as the time so spent will be well repaid later.

Titration with soap solution is unreliable, and, in the writer's opinion, should be discarded in favour of the acid methods described below.

Temporary Hardness.—The temporary hardness of the water is determined as explained in paragraph (*d*). On a 350 cc. assay the number of cubic centimetres of $\frac{N}{10}$ HCl required for exact neutrality give the result direct in degrees, or in terms of $CaCO_3$ per gallon. The temporary hardness is obtainable by inspection of the results of the complete analysis, being the sum in grains per gallon of the carbonates of calcium and magnesium present in the sample (the $MgCO_3$ must be expressed as $CaCO_3$). Conversely, a knowledge of the temporary hardness assists in the calculation of the amounts of those compounds existing in the water: hence its utility.

Total Hardness.—To the exactly neutralized liquid obtained by titrating with $\frac{N}{10}$ HCl in the temporary hardness determination, add 25 cc. of $\frac{N}{10}$ Na_2CO_3 and 25 cc. of $\frac{N}{10}$ NaHO. Boil down to about 150 cc. in a porcelain basin, filter, and wash with hot distilled water three times, cool and titrate to exact neutrality.

The effect of the Na_2CO_3 in the boiling solution is to precipitate all the salts of calcium as $CaCO_3$, which is only slightly soluble in water, and still less in solutions of alkaline carbonates. The NaHO precipitates $Mg(HO)_2$, which, although moderately soluble in water, is appreciably less so in solutions of alkaline hydroxides. For this reason considerable excess of both precipitating reagents is essential to the success of the assay.

The titration with $\frac{N}{10}$ HCl will determine the number of cubic centimetres of $\frac{N}{10}$ NaHO and Na_2CO_3 actually used in the precipitation. The number of cubic centimetres of alkali thus found

is equivalent (multiply by one) to the total Ca and Mg salts present, expressed in terms of $CaCO_3$ grains per gallon.

Permanent Hardness.—Deduct the temporary hardness, due to the carbonates of Ca and Mg, from the total hardness. The result gives the permanent hardness due to the sulphates and chlorides of those elements. If the permanent hardness thus obtained is expressed in terms of $CaCO_3$, the result may be reduced to CO_3 or CO_2 by multiplying by the factors given in paragraph (*d*);

or reduced to	$MgSO_4$	by multiplying	by	1.2
,, ,, ,,	$CaSO_4$	,, ,,	,,	1.36
,, ,, ,,	$MgCl_2$	,, ,,	,,	0.95
,, ,, ,,	$CaCl_2$	,, ,,	,,	1.11

As in the case of the temporary hardness, the permanent hardness is obtainable by inspection of the final results, and, conversely, a knowledge of its amount is an assistance in computing the probable composition of the salts in the water. For instance, working on the example illustrated in paragraph (*l*), the temporary hardness should agree with 7.48 gr. of mixed $CaCO_3$ and $MgCO_3$ expressed in degrees, which is the same thing as computing the result in terms of $CaCO_3$ in grains per gallon, and the permanent hardness should balance with 14.2 gr., being the total of the amounts of $CaSO_4$, $CaCl_2$, $MgCl_2$, and $MgSO_4$ expressed in degrees.

(*k*) Interpretation of Results.

According to the generally accepted theory of solution, the salts are present in the dissociated state, the ions being free; but, except where interaction has taken place, the compounds will occur in the water in those proportions in which they were originally dissolved during percolation through or over the rocks of the earth's crust, and it is the object of the assayer to distribute the acid radicals and bases to one another in such a manner as will be representative of the sample, being guided in his efforts by the total weight of the solids and the phenomena noted during analysis. It is sometimes possible to apply rules

for the apportioning of the acids to the bases, or vice versa, but these rules, which are based upon the relative solubilities of the various salts, cannot be applied universally. The re-combination of the results of an analysis of a complex unknown water is often difficult, and the amount of labour involved will depend upon the assayer's powers of observation, knowledge of the principles of chemistry, and ability to make rapid trial and error calculations.

In spite of the somewhat discouraging attitude adopted towards this branch of analysis by many books, the writer is positive that in most examples of mineral waters it is possible to determine very nearly the actual composition of the salts as they exist in the sample. A few of the phenomena which assist in the interpretation of results will be here given.

Carbonates of the Alkalies.—If the sample exhibits any permanent hardness neither Na_2CO_3 nor K_2CO_3 is present, and these compounds may be omitted from the calculations, for if introduced into the water they would interact with the sulphates and chlorides of calcium and magnesium.

Carbonates of the Alkaline Earths.—Boiling and concentrating will precipitate much of the $CaCO_3$ and $MgCO_3$ present. When waters are highly charged with these minerals an analysis of this precipitate will be of service, affording an approximation of the amounts of these compounds present in the sample, and reducing the final calculations.

Chlorides of the Alkaline Earths.—Advantage may be taken of the selective solubility of $CaCl_2$ and $MgCl_2$ in a mixture of alcohol and ether to effect a fairly good (sufficient for the purpose) separation of these compounds from all other salts of those metals usually met with in mineral waters.

Alkaline Silicates.—The presence of K_2SiO_3 in mineral waters is rare, and the small amounts of silicic acid determined, as described in paragraph (*b*), may usually be directly apportioned to the sodium base.

Water-Soluble Compounds of the Alkaline Earths.—The extraction of portion of the total solids with distilled water, and assay for either Mg or Ca, or perhaps both, will sometimes be worth

while, and will minimize much of the subsequent calculation. The foregoing remarks apply, of course, to fairly complex waters, and not to simple cases which do not require this treatment.

In order to have sufficient total solids available for carrying out any of the experiments mentioned, it is always well, when handling a new water, to evaporate down to dryness a considerable bulk of the sample, reserving the residue for confirmatory tests.

(*l*) Preparation of a Typical Mineral Water.

In order to illustrate the accuracy of results obtainable by the methods described in paragraphs (*a*) to (*i*), and to serve as an example of methods of interpreting results, a typical mineral water was very carefully prepared, the composition of which was accurately known. The salts were specially prepared for the purpose, and the composition of the sample was in accordance with the following table:—

Material weighed.	Quantity weighed.	Grains of anhydrous salt per gal.	Expressed as $CaCO_3$ in grm. per gal.
1. $CaCO_3$	0.2 grm.	3.08	3.08
2. $CaCl_2,6H_2O$	0.5 grm.	3.90	3.51
3. $CaSO_4,2H_2O$	0.5 grm.	6.10	4.45
4. $2MgCO_3,MgO$	0.2 grm., dissolved in CO_2 water	3.73 $MgCO_3$	4.40
5. $MgSO_4,7H_2O$	1.0 grm.	7.52	6.24
6. NaCl	1.2 grm.	18.52	
		Total	21.68
7. $Na_2SO_4,10H_2O$	0.7 grm.	4.76	
8. Na_2SiO_3 solution	equiv. to 0.21 grm. SiO_2	6.58	
9. KCl	0.5 grm.	7.71	
	Total	61.90	

To facilitate the preparation of samples and standards of definitely known composition, the following appended table of solubilities, &c., has been compiled, giving formulæ, water of

crystallization, and solubility of those substances either made use of as reagents or found when testing mineral waters.

For ready reference a table is also appended giving atomic weights of elements of importance in water analysis.

(*m*) Actual Example.

The sample illustrated on p. 93 gave the following results by the methods described :—

Grains per gallon.

	Ca.	Mg.	Na.	K.	Cl.	SO_4.	CO_3.	SiO_3.	Total Solids.
Sample as prepared	4.43	2.56	11.3	4.0	17.42	13.54	4.52	4.10	61.9
Results obtained	4.50	2.56	11.1	4.2	17.39	13.49	4.61	4.21	62.1

It now remains to apportion the acid radicals to the bases, making use of the weight of the total solids as a check on the result.

It is an advantage to commence the apportionment with those groups which are least likely to require much adjustment, dealing with those which may require several trial and error calculations later.

The SiO_3 may be distributed to the Na, 4.2 SiO_3 absorbing 2.5 Na, forming 6.7 Na_2SiO_3, *vide* the following table, which shows the result of each operation. Extraction of $CaCl_2$ and $MgCl_2$ by means of the solvent referred to in parargaph (*k*) yielded on assay $MgCl_2$ *nil*, $CaCl_2$ 3.7 gr. per gallon ; adding 10 % for incomplete extraction (which experiment indicates to be reasonable), the result is 4 gr. This reduces the amounts of Ca and Cl by 1.4 and 2.6 respectively, leaving 14.8 Cl to be apportioned between 4.2 K and 8.6 Na. As the Cl is in large excess, distribute the whole of the K to it (as a trial), forming 8.0 KCl, and reducing the amount of Cl by 3.8. Combining the remaining Cl with the only remaining base gives 18.1 NaCl, absorbing 7.1 Na and leaving 1.5 Na for combining with the only other possible acid—*i.e.* SO_4, giving 4.6 Na_2SO_4 and absorbing 3.1 SO_4. Extraction of the water-soluble compounds from the heated total solids yielded

on assay 1.4 Mg. As $MgCl_2$ was not present, this may be combined with SO_4, forming 7.0 $MgSO_4$. The still remaining Mg is now combined with CO_3, forming 4.0 $MgCO_3$. The remaining 1.8 CO_3 must be apportioned to Ca, forming 3.0 $CaCO_3$. The 1.9 Ca left should exactly balance with the 4.8 SO_4 remaining. This it does not quite do, as 1.9 Ca require only 4.56 SO_4 for saturation. Taking the 1.9 Ca as forming 6.4 $CaSO_4$, it will be seen that the total error is not very great.

Ca.	Mg.	Na.	K.	Cl.	SO_4.	CO_3.	SiO_3.
4.5	2.56	11.1	4.2	17.4	13.5	4.6	4.2
1.4	1.40	2.5	4.2	2.6	3.1	2.8	4.2
3.1	1.16	8.6		14.8	10.4	1.8	
1.2	1.16	7.1		3.8	5.6	1.8	
		1.5		11.0	4.8		
1.9		1.5		11.0	4.56		
1.9							
					0.2		

Na_2SiO_3 = 6.7.	$CaCl_2$ = 4.0.	$MgCl_2$ = *nil.*
KCl = 8.0.	NaCl = 18.1.	Na_2SO_4 = 4.6.
$MgSO_4$ = 7.0.	$MgCO_3$ = 4.0.	$CaCO_3$ = 3.0.
$CaSO_4$ = 6.4.		

Reporting Results.

When reporting results it is always well to quote figures obtained by direct analysis as well as the computed composition of the salts. In the case of the example illustrated the results would read as follows:—

Total solids at 200° C.	62.1	gr. per gallon.
Chlorine, Cl	17.4	,, ,,
Soda, Na_2O	29.9	,, ,,
Potash, K_2O	10.1	,, ,,
Magnesia, MgO	4.2	,, ,,
Lime, CaO	6.3	,, ,,
Silica, SiO_2	3.3	,, ,,
Sulphuric anhydride, SO_3	11.2	,, ,,
Carbon dioxide, CO_2	3.4	,, ,,

PROBABLE COMPOSITION :—

Calcium carbonate	3.0	gr. per gallon.
Calcium chloride	4.0	,, ,,
Calcium sulphate	6.4	,, ,,
Magnesium carbonate	4.0	,, ,,
Magnesium sulphate	7.0	,, ,,
Sodium chloride	18.1	,, ,,
Sodium sulphate	4.6	,, ,,
Sodium silicate	6.7	,, ,,
Potassium chloride	8.0	,, ,,
Total ..	61.8	

HARDNESS.

Temporary hardness	7.7°
Permanent hardness	14.1°
Total hardness ..	21.8°

TABLE GIVING SOLUBILITY, WATER OF CRYSTALLIZATION, &c.

Substance.	Formula.	Solubility at 70° Fahr. in gr. per gal.
Sodium chloride	$NaCl$	24,500
Sodium sulphate	$Na_2SO_4,10H_2O$	7,700
Sodium carbonate	$Na_2CO_3,10H_2O$	28,700 for crystals 8,400 for dehydrated powder
Sodium bicarbonate ..	$NaHCO_3$	6,860
Sodium thiosulphate ..	$Na_2S_2O_3,5H_2O$	35,000
Sodium hydrogen phosphate	$Na_2HPO_4,12H_2O$	10,500
Potassium chloride	KCl	23,800
Potassium sulphate ..	K_2SO_4	7,000
Potassium carbonate ..	$K_2CO_3,3H_2O$ ..	56,000
Potassium nitrate	KNO_3	21,000

Calcium oxide ..	.. CaO	70
Calcium hydroxide ..	.. $Ca(HO)_2$ ..	93
Calcium carbonate ..	.. $CaCO_3$	2.5
Calcium bicarbonate	.. $CaH_2(CO_3)_2$ ..	60
Calcium chloride ..	.. $CaCl_2.6H_2O$ ..	40 lb.
Calcium sulphate ..	.. $CaSO_4.2H_2O$ ..	161 gr.
Magnesium hydroxide	.. $Mg(HO)_2$ ..	2
Magnesium oxide ..	.. MgO	1.4
Magnesium carbonate	.. $MgCO_3$	1.5
Magnesium bicarbonate	.. $MgH_2(CO_3)_2$ ..	50
Magnesium chloride	.. $MgCl_2,6H_2O$ ..	20 lb.
Magnesium sulphate	.. $MgSO_4,7H_2O$ ..	24,500 gr.
Ammonium oxalate	.. $(NH_4)_2C_2O_4,2H_2O$	3,150
Ferrous sulphate ..	.. $FeSO_4,7H_2O$ ..	14,000
Barium chloride ..	.. $BaCl_2,2H_2O$ ..	24,500
Barium hydroxide ..	.. $Ba(HO)_2,8H_2O$	3,500

Water of Crystallization.

Ammonium magnesium phosphate ..	$NH_4MgPO_4.6H_2O$
Ammonium magnesium arsenate ..	$NH_4MgAsO_4,6H_2O$
Ferrous ammonium sulphate	$(NH_4)_2SO_4,FeSO_4.6H_2O$
Ammonium chloride	NH_4Cl
Silver nitrate	$AgNO_3$
Potassium iodide	KI
Potassium chromate	K_2CrO_4
Ferrous sulphate	$FeSO_4,7H_2O$
Magnesium sulphate	$MgSO_4,7H_2O$
Magnesium chloride	$MgCl_2,6H_2O$
Barium chloride	$BaCl_2.2H_2O$
Potassium bichromate	$K_2Cr_2O_7$
Plumbic acetate	$Pb(C_2H_3O_2)_2.3H_2O$
Sodium hydrogen arsenate	$Na_2HAsO_4,12H_2O$

ATOMIC WEIGHTS MADE USE OF IN WATER ANALYSIS.

As	75.0	Mg	24.0
Ba	137.0	Mn	55.0
Ca	40.0	N	14.0
C	12.0	O	16.0
Cl	35.5	Pt	195.0
Cu	63.0	K	39.0
Cr	52.0	Si	28.0
H	1.0	Ag	108.0
I	127.0	Na	23.0
Fe	56.0	S	32.0

THE ORGANIZATION AND EQUIPMENT OF A MINE-RESCUE STATION.

J. C. COLDHAM.

Discussion.

MR. HARTWELL CONDER wrote :—The author's description of the equipment of a mine-rescue station is full of interest and apart from its technical value serves to emphasize the need of every preparation against the contingency of fire. The writer had the opportunity of taking part in some of the rescue work on the occasion of the fire at North Mount Lyell and was especially concerned with animal tests for the presence of CO. The safeguards ensured by this means appeared so great that it seems to him that systematic provision should be made on those lines at every central station. A brief description of the methods arrived at may be of interest. The animals mentioned by Haldane in his "Methods of Air Analysis" were the mouse and small bird, such as canary or linnet. White mice were, he believed, employed on submarines. The children around Lyell did not seem to pursue the cult of the white mouse and cage birds were also scarce, so that when the writer wished to take some small animal into the mine, there seemed to be none available. The sight of some half grown fowls caused a start to be made with them and the descent and safe return of a young rooster from the main shaft helped in showing that the gases there were not too virulent. It was next found that week-old chickens could be obtained and a supply was secured. They were first used in cages, and the withdrawal of one practically moribund after five minutes at the bottom of the engine winze gave warning not to advance there too recklessly. As work went on better methods were evolved and use was made of the chickens in supporting the helmet men in

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exploratory work at the 850-ft. level. Work started from the engine winze and extended about 200 ft. along a level to the main shaft and beyond that to the main workings. The air at the engine winze was sweet and clear but there was smoke in all the workings away from it. The helmet men started out to go to the far workings. The writer's party followed with a chicken for about 50 ft. without protection. The chicken had a heavy nut tied with short string to its leg tethering it, and was placed on a dry sack at the side of the level. The party then returned to the winze leaving the chicken there for three minutes. It was remarkable how long three minutes seemed under such conditions. At the end of three minutes the party returned and found the chicken sound. A second chicken was then placed some distance beyond and treated in the same way. That chicken was also sound at the end of three minutes and a third was then placed beyond the main shaft. That little fellow gave signs of poisoning at the end of the period, but the helmet men had finished their work by then and returned, one of them only just able to walk. The party had been able to support the helmet men for quite a considerable distance under conditions of security impossible without the indications given by the small birds. The special advantage of the chickens lay first in their tameness. They could be handled without producing very abnormal conditions of circulation such as would develop in a cage bird or mouse. Secondly by holding the chicken in the hand very slight poisoning could be detected. The gas affected their legs just as it does those of men and the limpness of the little legs was at once indicated to the supporting hand. Under ordinary conditions of exploration the writer believes it would be safe simply to carry the chicken resting in the hand, being ready to retreat at the first sign of weakness. Incidentally it may be remarked that it was most difficult to make the men realise the usefulness of this test. Later on, at the repair work in the mine, it was suggested that some shaftmen should use the birds. While arrangements were being made for this one morning, word came up from the shaftmen who had slipped down below, that it was quite safe to send

the chickens down as the air was pure. One other recommendation regarding the work was that the men should be informed and warned as to the awful deadliness of the CO gas, not only immediately but in its after effects as well. If at North Lyell the gas in the later stages had retained or resumed the deadly character of the first six hours, there must have been some terrible fatalities in the rescue parties. One man showed the writer a sponge of vinegar which he used in front of his mouth. With this to protect him, on his own initiative he had gone exploring in stopes full of smoke, but fortunately for him clear of the more deadly gas. Others ran similar risks with no definite objective.

Papers and Discussions.

CONTENTS.

CYCLOMETER SURVEYS.

By LESLIE H. OWER.

FOR rapidly marking in geological and topographical features on parish plans, the author has used the Veeder cyclometer with very satisfactory results.

A simple Veeder cyclometer, usually retailed at cycle depots at 4s., is fixed to the axle of the front wheel of an ordinary bicycle. The cyclometer is designed to read in miles and tenths of miles, which are recorded through gearing actuated by a striker on a spoke of the bicycle wheel engaging one of the vanes of the cyclometer once every revolution.

By using 80 strikers the reading would be in chains and tenths of chains; therefore the use of 8 strikers gives the reading in chains, utilizing all the figures.

The cyclometer is designed to be placed on the right-hand side of the bicycle, and takes up a minimum of space, being only about the size of the top joint of the thumb. With some makes of cycles there is insufficient clearance, so the cyclometer must then be placed on the left side, and the 8 strikers made to engage the top vanes, as shown in the sketch (Fig. 1).

Very accurate readings can be obtained by this instrument. The author has found by numerous tests that the maximum error

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with $1\frac{1}{2}''$ tires is 2 % on the low side, obtained when leading the bicycle with tires pumped hard. The depression of the tire when riding reduces the radius, thus reducing the cyclometer error. As no allowance is made for grades, and it is often difficult to maintain a straight line, the cyclometer readings may, for ordinary purposes, be taken as quite correct.

No harm can be done by making the cyclometer read backwards. Readings to fractions of a chain can be taken, if desired,

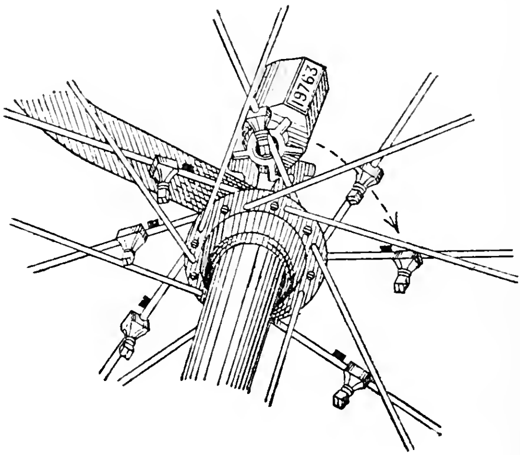

Fig. 1.
Cyclometer.

but for picking up geological boundaries and physical features, to be afterwards plotted on a scale of 40 ch. to 1 in., the nearest chain is sufficient, especially when short offsets are determined by eye and parallel traverses are anything up to 40 ch. apart.

The only error that can occur is through an accident to a striker or vane. This will be noticed at once when checking the readings on the parish plan, but, as they will maintain a definite percentage on the low side, the true chainage can be calculated. The author has recorded to date nearly a quarter of a million chains, and only once has he had any error due to this cause. It is a good idea to give an occasional glance to see that each striker

is engaging a different vane. Clerical errors also come to light when checking on the plans, but as the chainage is continuous the error is usually obvious, and can be corrected.

A method of recording observations in the field book, so that they can be at any time quickly referred to, is essential. The author has found an ordinary No. 16 level-book very satisfactory, the headings of the columns being altered, as shown in table A. A system of continuous chainage is used, which is recorded on the extreme left of the field book, and is very handy for index purposes. The time taken to record the cyclometer, the direction since the last reading, the class of country, the time, the aneroid height, the locality, and to offset by eye any physical feature, occupies on an average 30 seconds. Readings are taken whenever there is a change in the direction of the traverse, and, as the chainage never ceases, it is unnecessary to consult plans when in the field. A wrist watch supplies the time, which is utilized for adjusting the aneroid readings when working out reduced levels.

An aneroid that will record differences of altitudes of 10 ft. is carried in the right-hand coat pocket, and no attempt is made to read nearer than 10 ft. When commencing the day's work the aneroid is set as near as possible to the known datum, and is not corrected again during that day. All cross roads, and sometimes the main allotment corners, are recorded, both for plotting and to serve as aneroid stations when cross traverses are afterwards run. Directions of gullies and junctions of geological formations are determined by eye, readings taken at crests of rises, the positions of all points of high ground are indicated, and the height above the point where the reading is taken is measured by eye, or by pacing and aneroid if considered necessary. Every time an entry is made in the field book the aneroid is recorded, even though on level country. Roads and fences are followed, when available, and traverses at other than right angles are avoided if possible. The geology is best determined by uphill traverses, but observations made on down grades are frequently quite satisfactory.

The best method of commencing the survey of an area is to traverse a main road as far as time permits, and continue the observation along another road when returning home. Thus, in the process of mapping in the outer districts, aneroid stations and much information are obtained ready for use in the survey of the inner area.

When an opportunity occurs, the field book is written up in ink to form a permanent record. While doing this, a parish plan, mounted in folding pocket sections for preference, is constantly referred to. The rough bearings recorded in the field are altered to those mentioned on the plan, and the cyclometer chainage at cross roads and main allotment corners is written at the proper position on the plan. At these points, where traverses cross or overlap, each cyclometer reading is marked on the plan and cross-indexed in the field book, so that by using the plan as an index any traverse can be turned up in the field books in a few seconds. Such temporary references as "X roads" or "fence goes S." become "X roads at S.W. corner allot. — sec. —," or "N.W. corner allot.—" When inking in the aneroid heights an effort is made to reduce the error when possible, but experience shows that to spread the error over the whole day will not produce good results.

The reduced-level column is only filled in after linking up cross traverses prior to plotting the contours on the plan.

Table A gives an example of how notes are made in the field, and also shows how a new cyclometer can be inserted without interfering with the continuous-chainage system, only the last three figures of the cyclometer being used, the three figures on the left being continued for index purposes.

Table B gives the same field notes as written up after consulting parish plans in the office.

When plotting, measurements are usually taken from the nearest cross roads or a definite allotment corner, using the last two figures of the continuous chainage in preference to the subtracted back-sights.

Cyclometer surveying is also of great service when selecting sites for alluvial boring. Having decided to bore along a

certain line, generally a road, levels are run in the ordinary manner to secure a datum at one end of the line. The cyclometer is there recorded, and the assistant, according to the nature of the surface, rides, cycles, or walks a convenient distance ahead to suit the range of the dumpy level. The surveyor, while proceeding from one instrument station to the next, enters the cyclometer reading in the distance column as he passes the staffsman. Thus an approximate section is rapidly obtained, sufficiently accurate for selecting sites for bores from head office. Sights are taken on to the tops of chock pins of the angle stays at corner posts, which serve as bench-marks for obtaining the exact surface-level of each bore when the line is completed.

Table C gives the extracts from the field book for the survey of the N.E. corner of the parish of Merrymbuela (Ararat district).

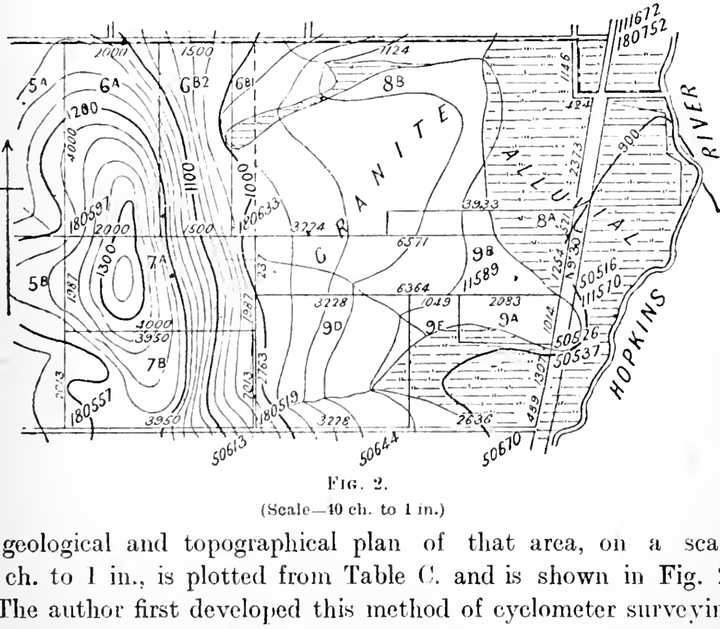

FIG. 2.

(Scale—40 ch. to 1 in.)

A geological and topographical plan of that area, on a scale 40 ch. to 1 in., is plotted from Table C. and is shown in Fig. 2.

The author first developed this method of cyclometer surveying in the Ararat district, where an area of 200 sq. miles, necessitating 2000 miles of cycling and 3000 aneroid readings, was geologically and topographically surveyed at the rate of two square miles per day.

TABLE A.

Distance by Cyclometer.	Back Sight.	*Inter. Sight. | * Fore Sight. Geology.	*Rise. Time.	*Fall. Aneroid.	Reduced Level.	Remarks. Cold and Squally. {Parish of Tatyoon (N.W.).
8/8/15—						
137843		alluvial	3.8	980		Cross roads, State school. See —.
W.						
137847		alluvial, basalt	3.10	980		Junction N.W. and indefinite to S.E.
W.						
137857		basalt	3.16	1000		New cyclometer inserted.
138000			3.22	1010		
W.						
138017		basalt	3.25	1020	B.M.	Railway crossing 134/46.
W.						
138023		basalt (?)	3.28	1030		Road turns S.E.
S.E.						
138026		basalt, granite	3.33	1040		Junction indefinite. Fence goes S.
S.						
138046		granite	3.40	1170		Fence turns E. Crest of westerly spur.
S.W.						
138057		narrow alluvial	3.44	1010		Gully at right angles. Trends 70° to R.
S.W.						
138073		granite	3.56	1090		Cross roads. See —.

* Words in smaller type to be read as crossed out.

TABLE B.

Distance by Cyclometer.	Back Sight.	Geology.	Time.	Aneroid.	Reduced Level.	Remarks. 8/8/15. Cold and Squally. { Parish of Tatyoon (N.W.)
137843		alluvial	3.8	930		See 132578, 50403, 206166. Cross roads at S.E. corner allot. 27.
W.						
137847	4	alluvial, basalt	3.10	930		Junction N.W. and indefinite to S.E.
W.						
137857 and 138000	over 10	basalt	3.16 } 3.22 }	950		New cyclometer inserted.
W.						
138017	over 17	basalt	3.25	960	942.44	Railway crossing 134 miles 46 ch. at allot. 33.
W.						
138023	6	basalt	3.28	970		Road turns to 144°.
144°.						
138026	3	basalt, granite	3.33	980		N.W. corner allot. 33. Junction indefinite.
S.						
138046	20	granite	3.40	1110		S.W. corner allot. 33. Crest of westerly spur.
216° across.						
138057	11	narrow alluvial	3.44	950		Gully from 130° turns to 285°.
216° across.						
138073	16	granite	3.56	1030		Cross roads at S.W. corner 34. See 146392, 206517.

TABLE C.

Distance by Cyclometer.	Back Sight.	Geology.	Time.	Aneroid.	Reduced Level.	Remarks. Parish of Merrymbuela (N.E.).
10/2/15— 190°. 50516	—	alluvial, granite	2.2	900	900	Stone culvert at N.E. corner 9A. See 11157.
190°. 50521	5	alluvial, granite	2.5	900	900	Granite ceases at E. side of road.
190°. 50526	5	granite	2.8	910	910	S.E. corner 9A.
190°. 50531	5	granite, alluvial	2.12	900	900	
10°. 50537	6	granite	2.20	910	910	S.E. corner 9A. Same as 50526.
W. 50548	11	granite, alluvial	2.23	910	910	Alluvial trending S.
W. 50567	19	alluvial	2.27	910	910	W. boundary allot. 9E.
W. 50584	17	hill wash	2.33	940	940	Rise now more pronounced.
W. 50599	15	granite	2.43	1030	1020	Granite exposed.

S. 50608	7	narrow alluvial	2.47	1000	990	Mountain gully trending E.
S. 50613	5	granite	2.50	1030	1020	E. and W. road at S.W. corner 9D. See 180519.
E. 50657	44	granite, alluvial	3.18	890	890	Junction N.W. and S.
E. 50670	13	alluvial	3.22	880	890	Cross roads N.E. corner allot. 10. See 111541.
14/5/15— 111541	—	alluvial	3.30	910	890	Cross roads N.E. corner, allot. 10. See 50670.
10°. 111570	29	alluvial, granite	3.37	920	900	N.E. corner 9A. See 50516.
W. 111589	19	granite	3.45	920	900	N.W. corner 9A.
N. 111606	17	granite	3.55	930	910	N. boundary 8A.
N. 111619	13	granite	3.58	950	930	Crest of rise. Alluvial 5 ch. to E.
N. 111638	19	narrow alluvial	4.7	930	910	Creek trending E.
N. 111641	3	granite	4.10	940	920	E. and W. road on N. boundary allot. 8B. See 114280.
E. 111648	7	granite, alluvial	4.13	930	910	
E. 111672	24	alluvial, basalt	4.18	930	910	Main Ararat–Willaura Road.

TABLE C.—*Continued.*

Distance by Cyclometer.	Back Sight.	Geology.	Time.	Aneroid.	Reduced Level.	Remarks. Parish of Merrymbuela (N.E.).
17/10/15— 180439	—	granite	9.35	920	920	S.E. corner 11B.
N. 180491	52	granite	9.58	1040	1040	Gully trending 95°.
N. 180504	13	granite	10.5	1180	1180	Crest of abrupt spur trending S.E.
N. 180519	15	granite	10.14	1010	1010	E. and W. road at S.W. corner 9D. See 50613.
W. 180538	19	granite	10.24	1180	1180	End of sharp rise and head of gully mentioned 50608.
W. 180547	9	granite	10.26	1210	1210	Plateau-level. Point 20 ft. higher 10 ch. N.E.
W. 180557	10	granite	10.30	1190	1190	S.W. corner 7B. Head of depression trending 250°.
N. 180565	8	granite	10.35	1200	1200	Crest of flat westerly spur.
N. 180571	6	granite	10.38	1180	1180	Head of depression trending W., then N.W.
N. 180585	14	granite	10.44	1250	1250	Crest of westerly spur. Highest point, 90 ft. higher, is 12 chs. to E.

N. 180597	12	granite	10.55	1200	1200	S.W. corner 6A. At 6 ch. to W. is head of depression trending 340°.
E. 180610	13	granite	11.0	1300	1300	Crest of ridge. Level for 6 ch. to N.
E. 180633	23	granite	11.12	1010	1010	S.E. corner $6B_2$. At 5 ch. to S. gully debouches from W.
N. 180641	8	granite	11.15	1010	1010	Crest of sharp N.E. spur. Sharp rise commences 6 ch. to W.
N. 180653	12	granite, alluvial	11.18	970	970	Junction N.E.
N. 180657	4	alluvial, granite	11.20	970	970	Junction E. Alluvial does not extend far west.
N. 180662	5	granite	11.24	990	990	Fence goes E. Crest of spur 10 ft. higher 6 ch. to N.
E. 180678	16	granite, alluvial	11.28	950	950	Granite 1 ch. to S. Alluvial due to gully from 180657.
E. 180684	6	narrow alluvial	11.30	930	930	Granite 10 ft. higher 5 ch. S. Fence goes N. Gully continues at 85°.
N. 180690	6	narrow alluvial	11.34	930	930	Short gully from W.
N. 180695	5	granite tor	11.38	970	970	E. and W. road. Highest point, 20 ft. higher 4 ch. N.
E. 180752	57	alluvial, basalt	11.45	910	910	See 11672. Cross roads. Ararat-Willaura Road.